# GOLD MILLING

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16

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AND

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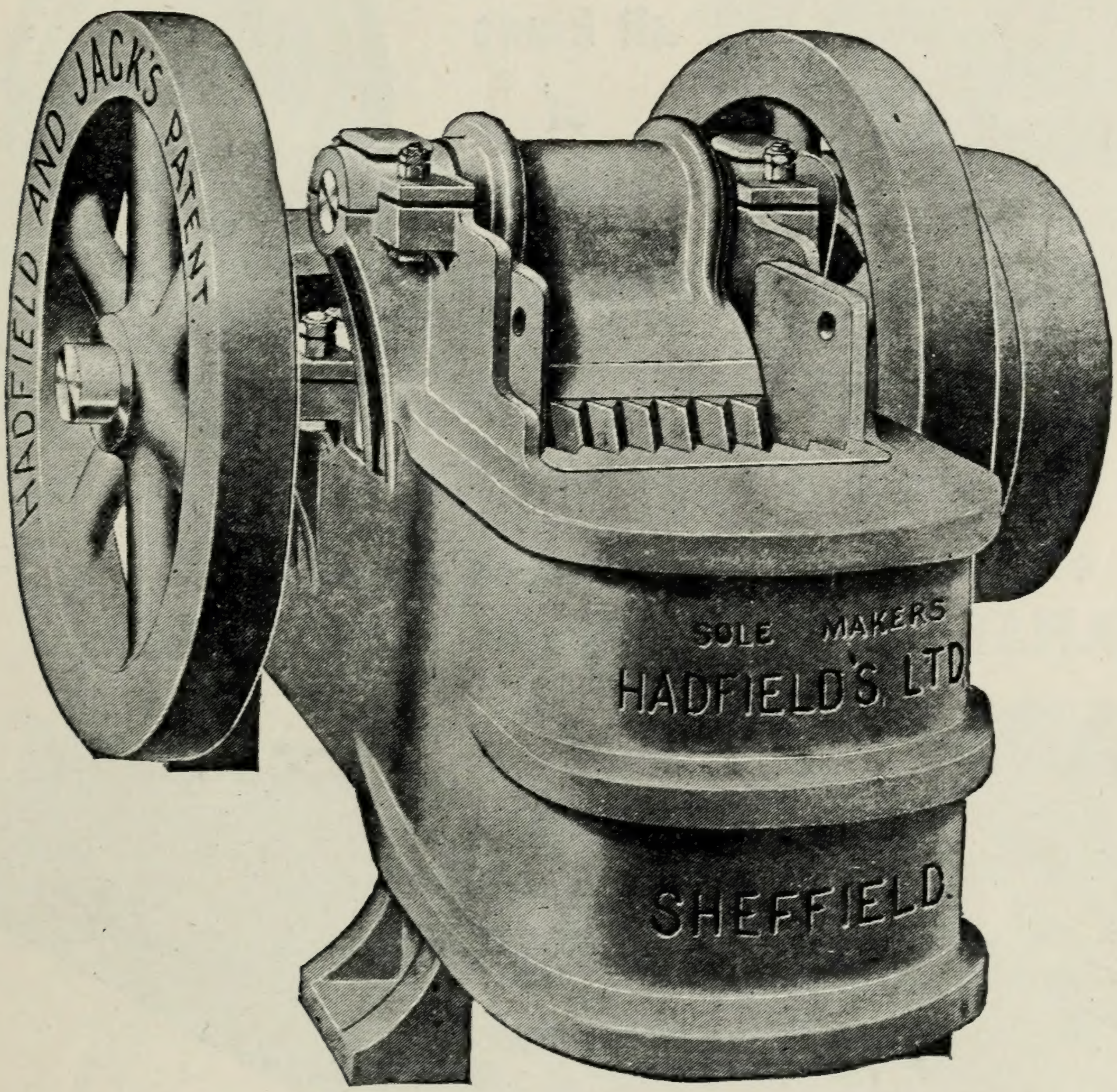


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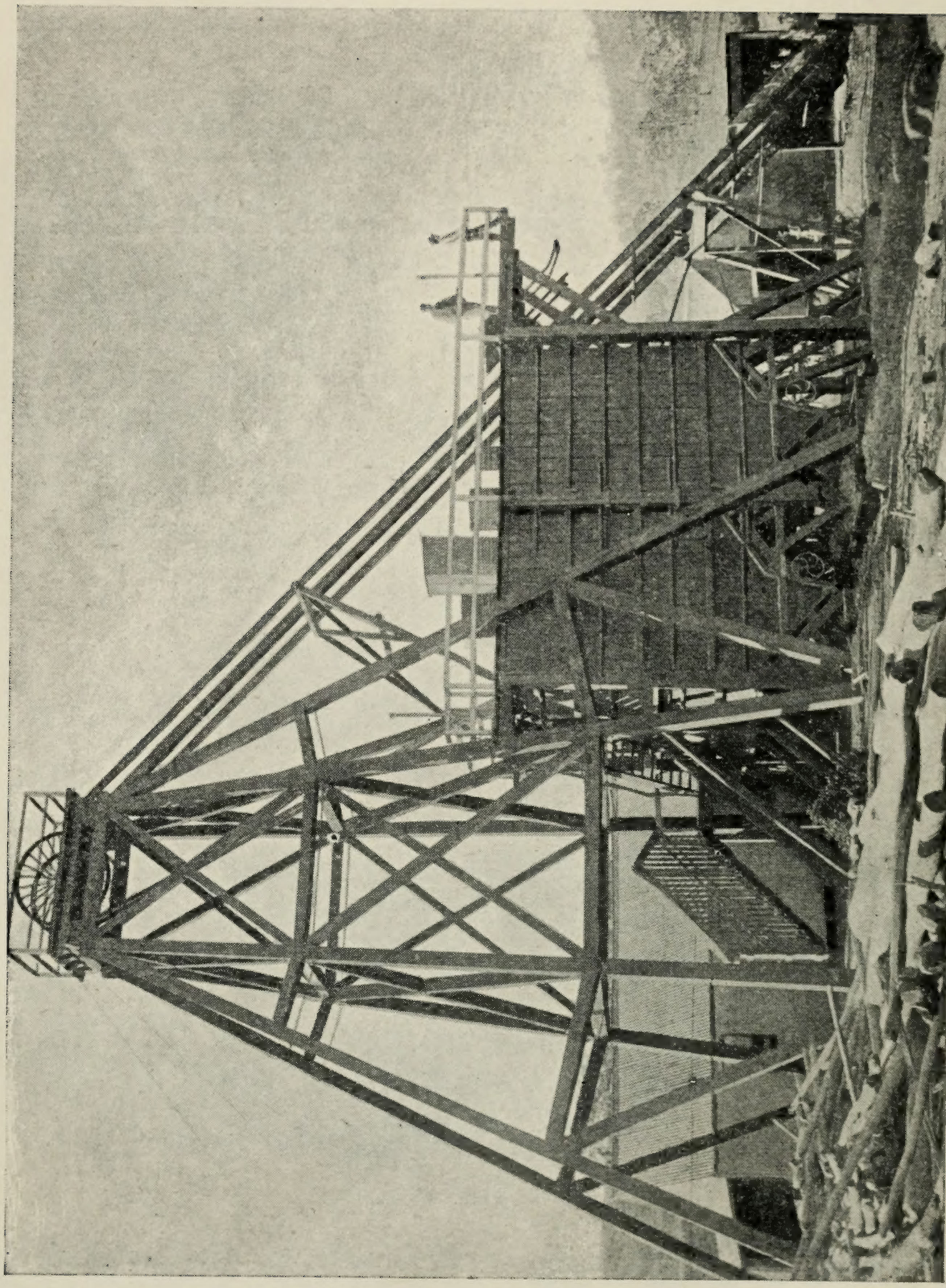

HEAD-GEAR AND BINS.

# GOLD MILLING

## PRINCIPLES AND PRACTICE

BREAKING; SORTING;
WET AND DRY MILLING; AMALGAMATION;
CONCENTRATION; ROASTING; CHLORINATION; CYANIDATION;
MELTING BULLION; SMELTING ORES AND CONCENTRATES; COMPLETE SYSTEMS, RESULTS, AND
COSTS; TABLES; AND FORMS.

Charles George
BY C. G. WARNFORD LOCK
F.G.S. M. INST. M.M.
LATE (1895-1899) SUPERINTENDENT OF THE WENTWORTH AND
ALADDIN MINES, NEW SOUTH WALES

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# PREFACE.

An attempt is here made to present, between one pair of covers, a comprehensive guide to the whole series of operations embraced in the industry of extracting gold from the various rocks and ores with which it is associated.

The book is written throughout from the practical standpoint of the mill manager and his workmen. Agricola and mediæval alchemists are given a well-earned rest, and attention is confined to modern practice and the demands of the conditions of to-day.

Underlying principles, both in mechanics and in chemistry, are explained when necessary in justifying the adoption or rejection of proposed new machines or methods, without, however, trespassing upon such mathematics or abstruse chemical problems as are outside the province of actual mill management.

The commercial side of the question is kept always in view, and much labour has been expended in reducing to one common denomination the maze of statistics contained in official reports, etc. For this purpose, the American or "short" ton of 2000 lb. has been adopted as the standard; it is already recognised in the United States, Canada and South Africa, and is incomparably the most simple for calculation.

In the form given, the figures should prove not merely of idle interest but of actual utility, suggesting to managers

where their methods are open to amendment, and placing in the hands of directors a key to criticism of cost sheets.

Some useful tables and ideal forms for returns are included.

To an early training in chemical works, and many years' personal management of amalgamation, chlorination and cyanidation plants, the Author has added such information as he could gather from visits to other mills and from published records of the researches and procedure of other managers, the sources having been, it is hoped, acknowledged in all cases.

Conscious that errors both of omission and commission must necessarily have crept into his pages, through the changes of procedure that are constantly being demanded in such a progressive industry, the Author will most gladly avail himself of just criticisms or corrections, and be pleased to acknowledge their source in future editions.

C. G. WARNFORD LOCK.

c/o The Institution of Mining and Metallurgy,
Broad Street House, London, E.C.
*January, 1901.*

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# GOLD MILLING.

## CHAPTER I.

### STORING, SCREENING AND SORTING.

**Shaft-head Bins.**—A prime factor in keeping down milling costs and maintaining regular outputs is a constant supply of ore to the mill. This cannot be counted on unless provision is made for accommodating on surface at least as much ore as the mill can put through weekly. Various circumstances tend to render the raising of ore from most mines anything but regular throughout the day or from day to day. Generally some waste or mullock has to be brought to surface as well as ore, and this must usually—unless special winding plant is provided for it—cause a suspension of ore-raising, rendering it intermittent. Then again, with small and irregular ore bodies, waste rock may considerably preponderate at times, even to such an extent as to monopolise the hoisting-engine for a day or two; while at other times the ore may be broken down at a much greater rate than the mill can take it, and stowage underground may be inconvenient, and entail additional handling. Moreover, an accident, such as all machinery is liable to, may actually prevent hoisting for some days, as also may repairs and alterations to shafts. All these contingencies point to the

desirability of a substantial reserve of ore being binned above ground.

No better site can be found for a main reserve bin than at the shaft head. The timbers of the head-gear can give material assistance in the support of the bin, and tipping-gear on the brace enables the truck or skip to be quickly emptied and sent below again.

The practice on the Rand, which is the most advanced

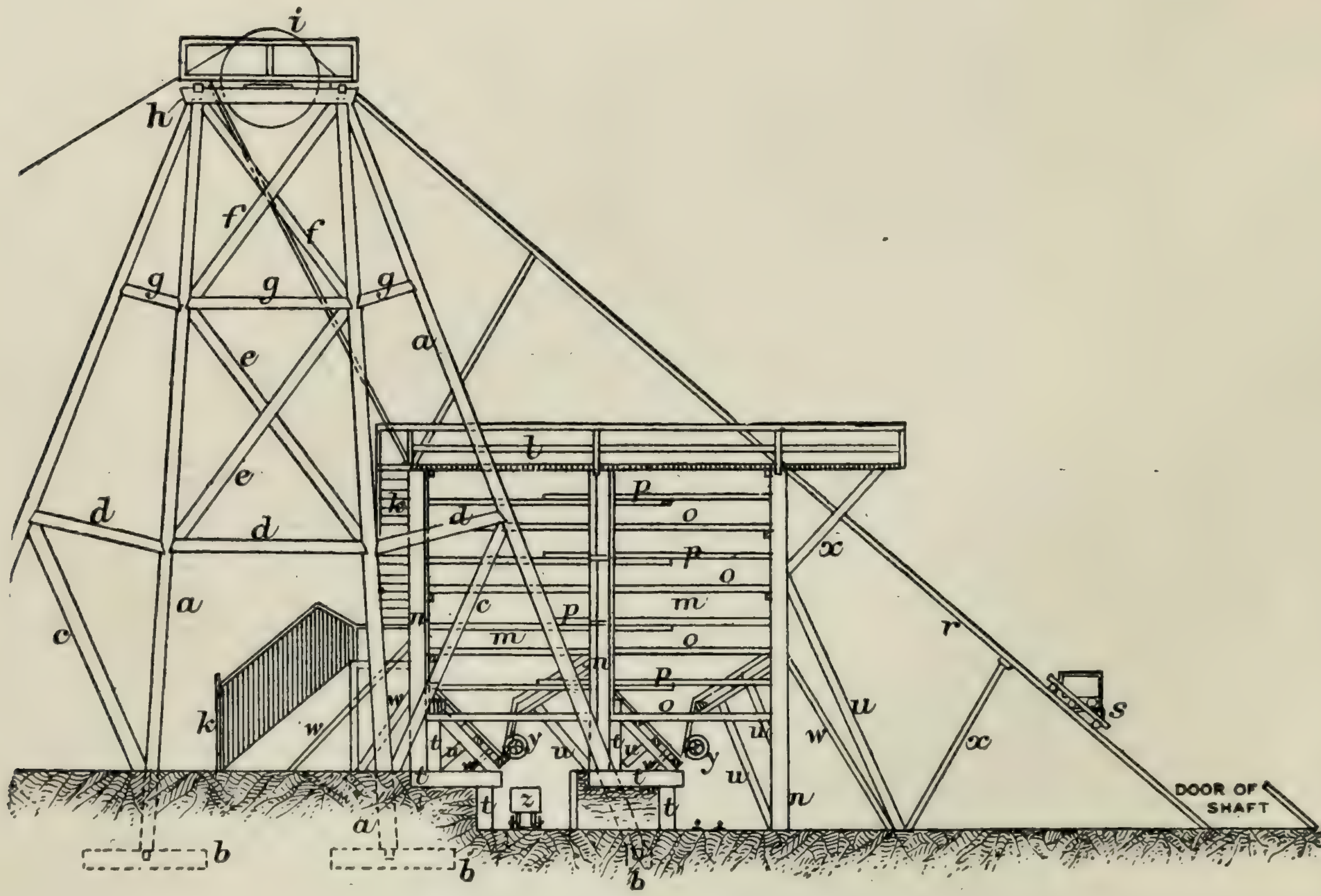



FIG. 1.—HEAD-GEAR AND BINS.

of all gold mining fields, especially in the cheap and rapid handling of large quantities of ore,* is to equip the head-gear with a separate bin for waste or mullock of say 100 t. capacity, and with double bins for the ore. This latter is tipped on a grizzly (see p. 10), which allows the fines to pass at once to the main bin, while the coarse is conducted to a picking-floor or table, where large pieces of waste are removed

* See Hatch and Chalmers, 'Gold Mines of the Rand,' and S. J. Truscott, 'Witwatersrand Goldfields.'

by hand, and the pay stone is passed on to the mouth of a breaker which discharges into the main bin, having a capacity ranging from 150 to 1000 t., but commonly about 200 to 300 t. In some mills, a grizzly only is placed in the head-gear to remove fines, and the sorting and breaking of coarse ore proceed in a separate building, or in the battery itself.

A general view of shaft-head bins where no screening or sorting is done is given in the frontispiece; it illustrates the arrangement at the Phœnix inclined shaft of the Wentworth Proprietary Co., Lucknow, N.S.W., Australia, erected under the author's direction. Figs. 1 and 2 show the details of

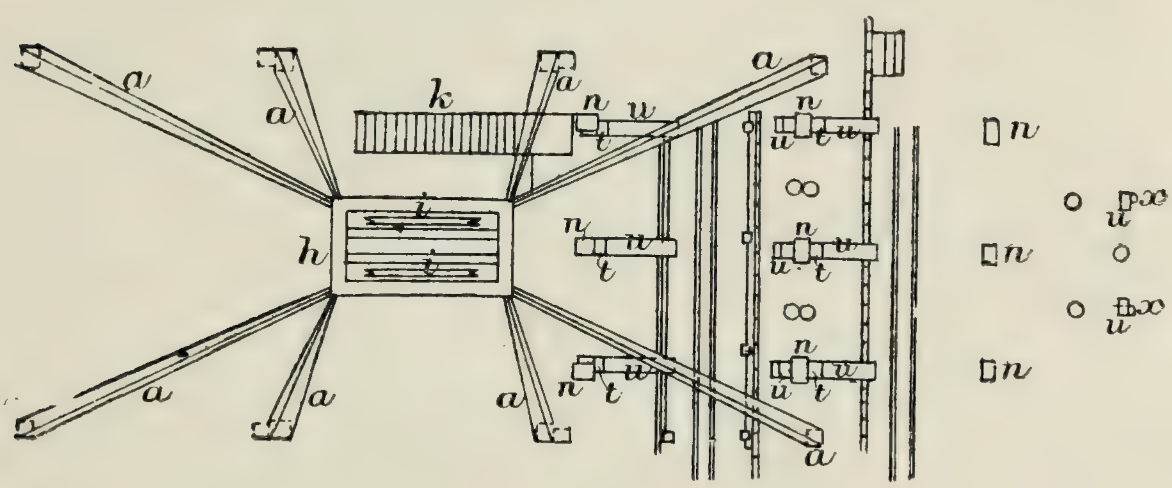



FIG. 2.—HEAD-GEAR AND BINS.

construction in side elevation and horizontal section respectively. The poppet legs *a* are of selected Oregon pine, 14 in. sq.; they are sunk 8 ft. in the ground, and are mortised into ironbark bed logs *b*, 18 in. sq. and 10 ft. long. The four struts *c* are 12 in. sq. The cross stays *d* are also 12 by 12 in.; they are stump-mortised, and held secure by 1 in. round iron rods, with nuts at the ends, and iron plates to prevent cutting into the timber. The angle ties *e* are 12 by 8 in., and *f* are 10 in. sq.; the cross-stays *g* are also 10 by 10 in. The top frame *h* is made of 14 by 10 in. stuff. The sheaves *i* have a 4 ft. radius. A stairway *k*, the lower portion of which is close-latticed and provided with a locked door to prevent access of children and unauthorised persons, leads to the brace *l*, surrounded by a very substantial railing. The

bins *m* are in duplicate, one receiving ore from the Wentworth Co.'s mine, and the other from the workings of the Aladdin's Lamp Co., these sharing the use of the Phœnix shaft. Each bin has internal measurements of about 18 ft. in depth by 12 by 16 ft., and when full it will accommodate some 225 t. of milling dirt. They are constructed entirely of Colonial "hardwood," embracing various kinds of eucalypt. The posts *n* are 12 in. sq., and are seated in substantial bed logs, the same as the poppet legs *a*. The wooden ties *o* are 6 in. sq., and are strengthened by intervening steel (old) railway rails *p*, which were obtained very cheaply from the Government railways. The inside planking is 12 by 2 in., and is lined

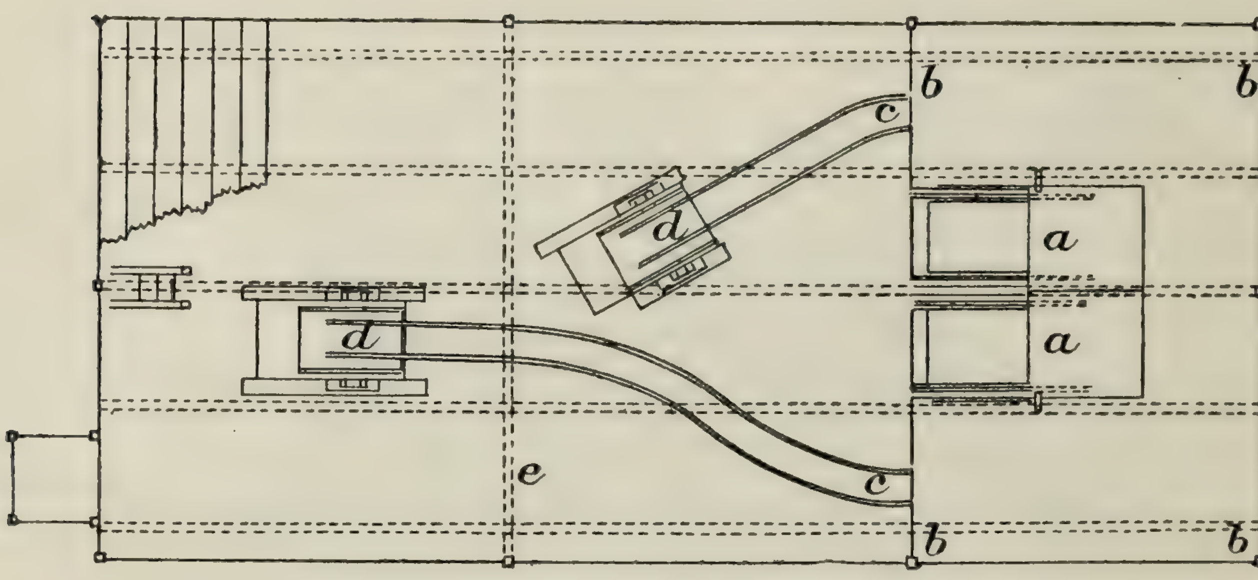



Fig. 3.—Head-Gear and Bins.

with $\frac{1}{8}$-in. steel sheets. The sheets in the lower portion were all new; but for the upper parts an economy was effected by using a number of discarded flat-sheets from the mine, some of which were not steel, and many of which did not exceed $\frac{1}{16}$-in. in thickness. The runners *r*, carrying the tram rails by which the truck-carriage *s* reaches the brace *l*, are 8 by 4 in. The underpinning of the bins consists of posts and sills *t* 12 in. sq., and struts *u* 10 by 8 in. Experience showed these to be insufficient, for after about two years' life, the ends of the bins began to bulge and the floor joists to sag. Supplementary supports *w* of 8 by 8 in. hardwood were then added, and served satisfactorily. The struts *x*, 6 in. sq., support the tramway and the overhanging floor of the brace

respectively. The discharge-doors *y* deliver into the trucks *z* beneath. These latter are of a capacity to hold about one long ton each, and are drawn, three at a time, by a horse to

FIG. 4.—HEAD-GEAR AND BINS.

the breaker bins about 200 yards distant, the road being laid with 16-lb. steel rails.

The loaded mine-truck on *s* passes through a space in the floor of the brace *l*, which is then closed by a balanced swing door, as shown at *a* in Fig. 3. The floor of the brace is laid

with 12 by 2 in. hardwood planks, and the portion *b* around the doors *a* is covered with $\frac{1}{8}$-in. steel flat-sheets well screwed

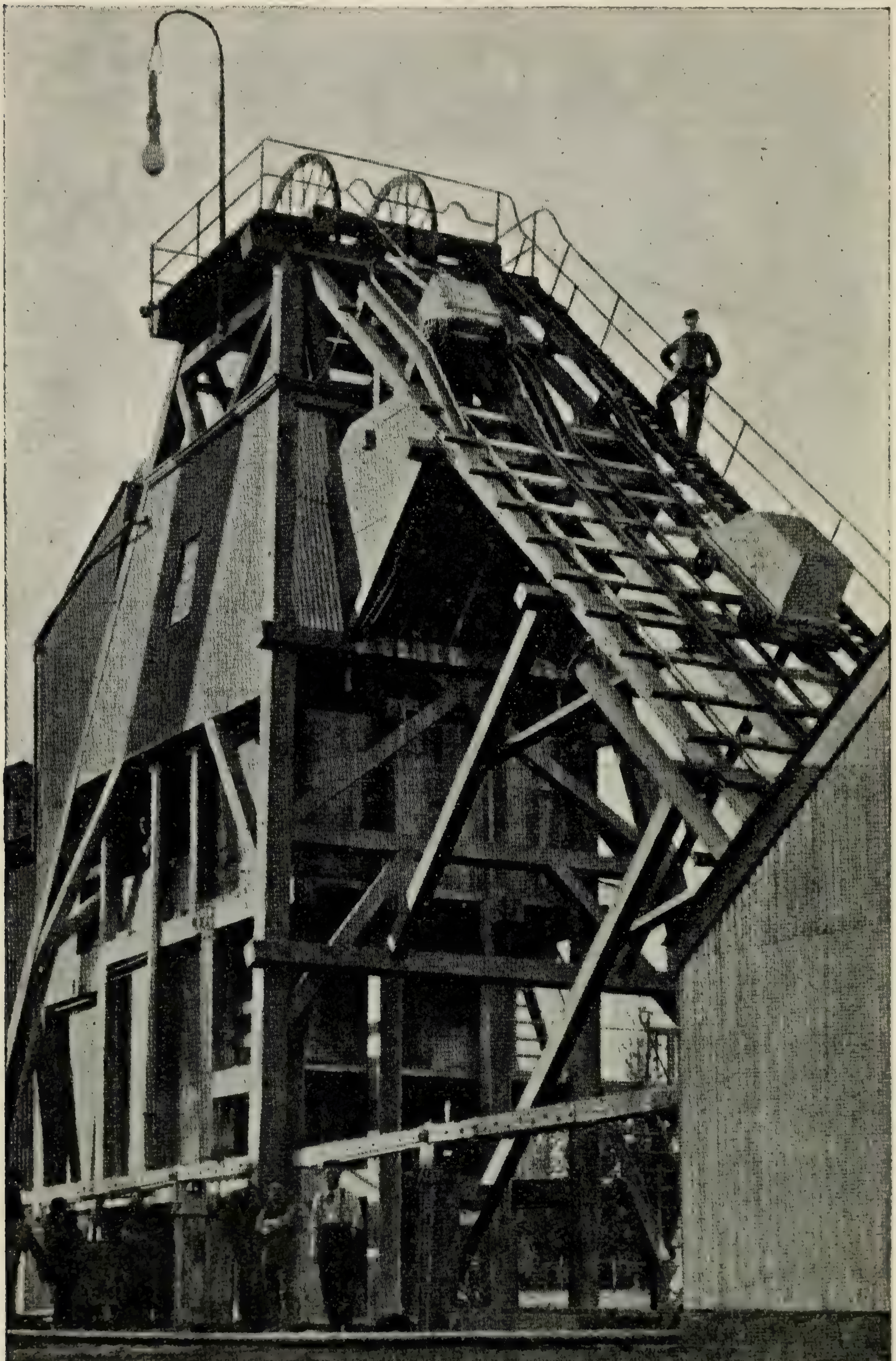


Fig. 5.—Head-Gear and Bins.

down, on which the truck, when disengaged from the carriage, can be readily guided to either set of rails *c*, leading to the tumblers *d*, one for each bin, the division being at *e*.

Examples of head-gear at which breaking and picking accommodation is provided are illustrated in Figs. 4 and 5, the former being erected on a vertical shaft, and the latter on an inclined shaft.

*Tumbler.*—The tumbler is shown in detail in Figs. 6, 7 and 8: *h* is a wooden foundation (6 by 4 in.) to which are secured by 1-in. bolts, the wrought-iron standards *c*, carrying cast-iron bearings *b*, in which is suspended, by a wrought-iron fulcrum *a* on each side, the sheet-iron cradle *d*. This last is of $\frac{1}{8}$-in. material, strengthened all round at *f* by a strip of 2 by $\frac{3}{8}$-in. flat iron, to take the rub of the truck, and at *e* by 2 by $\frac{1}{2}$-in. angle iron to carry the top rim *g* of $\frac{1}{4}$ in. flat iron. Counterpart rails (steel T rails 14 lb. per yd.) to those on the brace floor are laid in the bottom of the tumbler as at *i*, and are turned up at *k*, so that, when the truck is pushed home, the tumbler and truck at once describe a

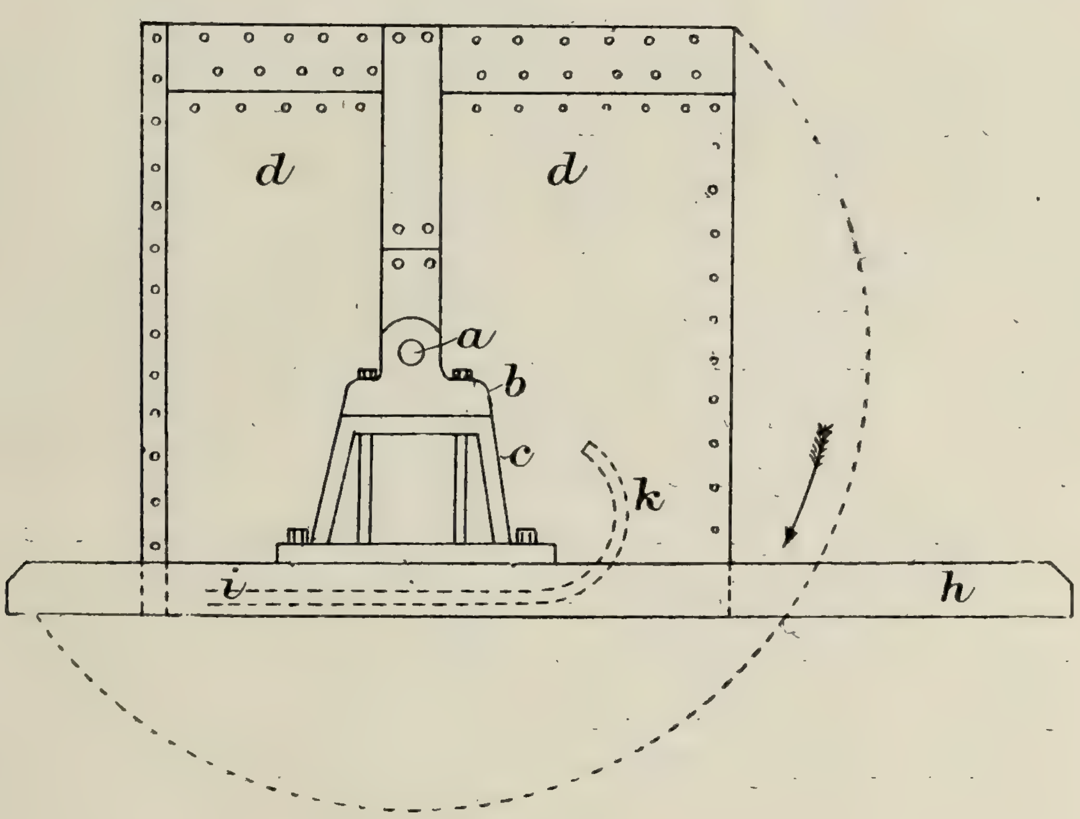



FIG. 6.—TUMBLER.

semicircle, thus dumping the contents of the truck, while the impetus gained brings the empty truck back into its upright

position. The arrangement proved most satisfactory, making the work simple and expeditious, and entailing no repairs whatever.

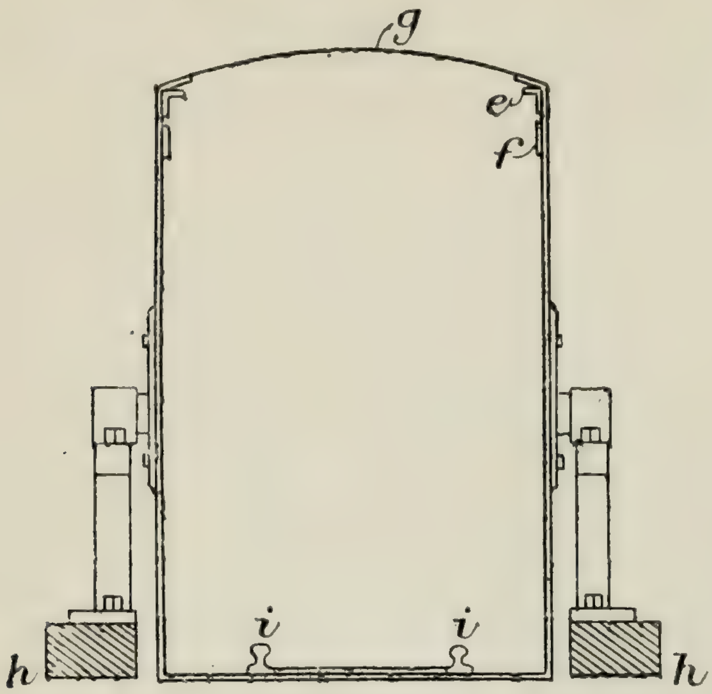



FIG. 7.—TUMBLER.

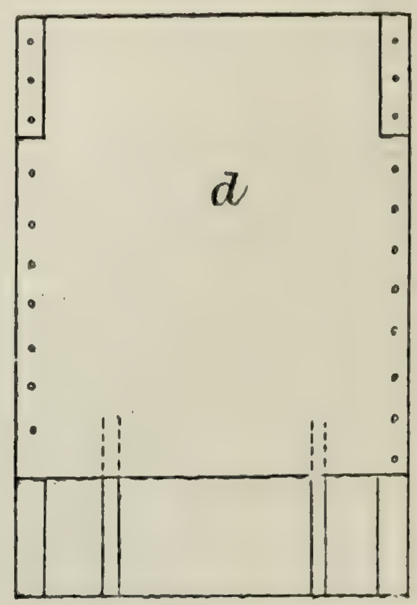



FIG. 8.—TUMBLER.

*Bin Doors.*—The method of constructing the floor of the bin, that is, with a shelf, as it were (see Fig. 1), is most successful in relieving the discharge-door from pressure, so much so that, even when a bin is quite full, a lad of 16 has no difficulty at all in filling the trucks from it and completely controlling the issuing ore. As a general rule, the "dirt" is just dry, without being dusty, and will run freely at the angle shown, and only occasionally does it become necessary to use a bar to disturb packed boulders. Even then, with one hand occupied by the bar, the other can easily reach and govern the wheel *y*. The door, as in most of the shoots throughout the mine, is on the rack-and-pinion principle, and is shown in detail in Fig. 9. The door is of $\frac{1}{4}$-in. sheet iron, 3 ft. $5\frac{1}{4}$ in. wide and 2 ft. 3 in. deep; the front plate is 3 ft. $5\frac{1}{2}$ in. wide, 2 ft. $3\frac{1}{2}$ in. deep, and $\frac{3}{16}$ in. thick; the bottom plate is 3ft. $5\frac{1}{2}$ in. wide, and $\frac{3}{16}$ in. thick. The sides also are of $\frac{3}{16}$-in. iron. The frame timbers *a* are 10 in. sq. The lugs are $\frac{3}{4}$ in. thick; and the rack is secured by $\frac{5}{8}$-in. bolts, with countersunk heads flush on the inside. The pinion bearings are 14 by $3\frac{1}{2}$ in.; the pinion is $1\frac{3}{4}$ in. diam. with 1 in. pitch,

and has sixteen teeth 3 in. wide. The wheel is 2 ft. diam., and its rim is $1\frac{3}{4}$ in. thick, with $\frac{3}{4}$ in. clearance from the bearing. The two top angle bars are secured by bolts and nuts to the sides, so that they can be unshipped to remove the door. The rack is 1 in. thick, with $\frac{3}{4}$-in. teeth.

Perhaps a more common form is a sliding door controlled by a long handle. This is simpler and less costly in construction, but it is far less under control. Moreover, the handle is often in the way, and, in case of the door jamming, it is very apt to get bent by strains. Should the ore pack in the shoot, as will happen sometimes, or any other obstruction occur, such as from drills carelessly dropped in, or from slabs getting loose in the sides of the bin, then a crowbar may have

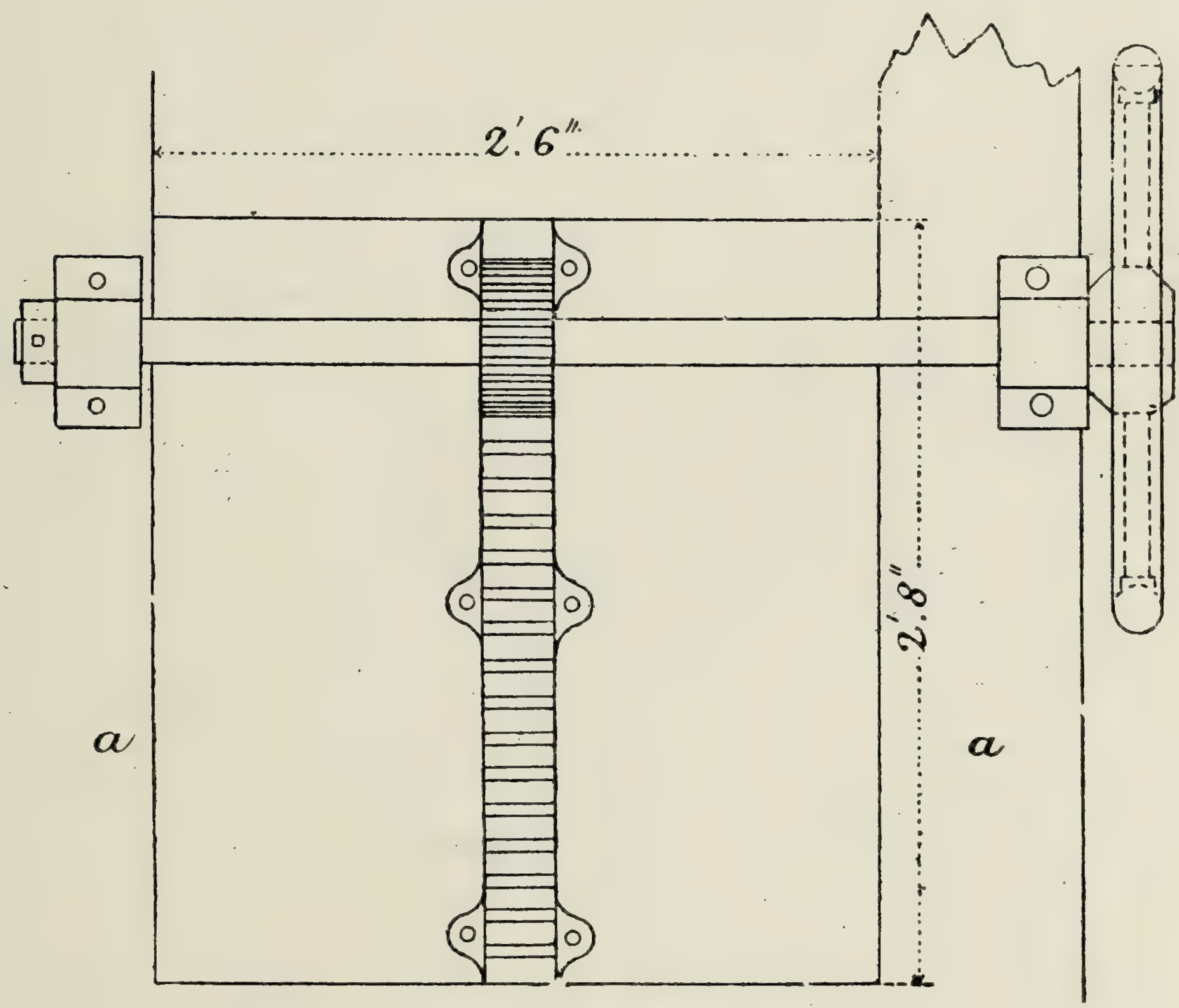



FIG. 9.—BIN DOOR.

to be used to remedy matters. It is very difficult to manage a crowbar effectually, and at the same time to control the lever, so that much mess and trouble may be caused by a

sudden rush of the loosened ore. Certainly the rack and pinion is much handier and more direct-acting. Occasionally the door is placed in the floor of the bin, instead of in the bin face, but this should be avoided if possible. For dry hard rock, an angle of 40° will often suffice for the floor, but it is safer to make it 45°, and indeed some soft and clayey ores will not run at less than 50°.

**Screening.**—The automatic separation of fines from the mass of ore before proceeding either to the picking-floor or to the breakers direct, is effected by fixed screens or grizzlies. These consist of a series of bars 10 to 15 ft. long, laid parallel to each other at an angle of about 45°, and at a distance apart varying from 1 in. to 2 in. They will occupy a width commensurate with the size of the truck or shoot delivering ore upon them. When made to order, these bars are best of steel, 2 in. deep, $\frac{3}{4}$ in. wide on the upper-face and $\frac{1}{2}$ in. on the under-face, so that packing may be avoided. They are kept apart by small cast-iron distance pieces, which are perforated, and held in place by tie rods passing through them and the bars. Old tramway rails are often used for the purpose of grizzlies, instead of new steel bars, and answer well if placed base uppermost. When the rock is not very hard, or it is going to be picked, too small screening is not advisable; but with really hard rock, great economy is gained in milling by reducing the ore as far as possible before it enters the stamps; and by setting the grizzly at a trifle less than 1 in. spaces an additional opportunity is given for arresting stray drill-heads and other foreign bodies, which will find their way into the dirt, and are apt to do damage in the battery. A 1-in. space is common on the Rand.

**Sorting.**—In every mine some degree of care is taken to separate waste from milling dirt, at any rate so far as the larger and more evident pieces are concerned, and at most establishments this is deemed sufficient. There are numerous instances where no further picking would be remunerative, both because of the cost and because of the liability to

reject valuable stone hidden in the waste. This is particularly the case at mines carrying high-grade ore in very small seams, permeating, it may be, a mass of barren rock. But on some fields, and especially in the Transvaal, precisely the opposite condition prevails, and there sorting above ground is carried to an extreme point.

It is not considered economical to pick anything which passes through the grizzlies, and their setting is regulated by that consideration. But on the Rand all ore which refuses to pass through the grizzlies is subjected to a careful picking on floors, rotating tables, or travelling belts. In a few cases, the sorting is carried out in two stages, a first grizzly being set at 3 in., and its rejections being picked for coarse waste before going through a breaker, and again afterwards. The aim of this operation is to throw out only clean waste each time, but that this ideal is not reached is shown by the figures given in the table on p. 13, from which it will be seen that the waste reaches as high as nearly 1¼ dwt. per ton. This would seem to indicate carelessness, resulting in some pay stone being sacrificed and helping to raise the general average. The loss is regrettable, but it is justified on the far-reaching demands of financial benefit. Thus if the value of the rejections is kept below that of the ultimate residues from the mill, and if the cost of sorting out waste is less than the cost of milling it after crediting the milling with the extraction gained, commercial considerations favour the practice, assuming that the mill can be kept fully employed on more profitable rock.

Thus, at the Bonanza mill, by rejecting 28½ % of waste, the grade of ore is raised from 19·55 dwt. at the sorting floor to 26·77 dwt. at the mill, which is equivalent to a rise of 30*s.* per ton at a cost of 1*s.* 3½*d.* The Paarl mill had accommodation only for picking out about 11½ %, but has now increased its crushing and sorting room so as to reject 30 %. Similarly at the Van Ryn mill, which has by degrees reached 22·7 %, it is estimated that not less than 30 % should be aimed at, and that this will raise the average yield of the mill by ½ dwt. per ton. The Ferreira Co.'s report contains a statement showing

the effects of sorting up to the end of 1898, from which the following has been prepared :—

| | Waste Rock Picked Out. | | Value of Ore Mined. | Value of Ore Milled. | Increased Value of Ore Milled due to Sorting. | | Value of Additional Gold Won. | Direct Saving due to Sorting. |
|---|---|---|---|---|---|---|---|---|
| | % | Assay dwt. | dwt. | dwt. | dwt. | % | £ | £ |
| 1893 | 15·60 | ·76 | 19·38 | 22·69 | 3·31 | 17·08 | 28,899 | 6,825 |
| 1893 | 31·19 | ·90 | 19·70 | 27·95 | 8·25 | 41·88 | 36,526 | 9,360 |
| 1894 | 34·36 | ·97 | 22·49 | 33·49 | 11·00 | 48·92 | 52,599 | 10,282 |
| 1894 | 44·72 | 1·06 | 19·33 | 33·76 | 14·43 | 74·66 | 109,547 | 14,656 |
| 1895 | 38·50 | ·96 | 19·42 | 30·97 | 11·55 | 59·49 | 150,405 | 22,966 |
| 1896 | 32·50 | 1·01 | 17·25 | 25·04 | 7·79 | 45·15 | 199,932 | 27,843 |
| 1897 | 32·95 | ·49 | 17·35 | 25·04 | 7·95 | 44·31 | 216,375 | 24,105 |
| 1898 | 40·11 | ·83 | 13·46 | 21·93 | 8·46 | 62·85 | 220,679 | 41,878 |
| | 35·02 | ·84 | 17·17 | 25·78 | 8·60 | 50·10 | 1,014,965 | 157,918 |

The total waste picked out reaches 308,189 t.

In nearly all statements of milling expenses, the figures relating to sorting are lumped together with those of tramming and breaking, so that it is impossible to arrive at the details, but in the following typical examples the costs are itemised :—

| — | Labour. | Fuel. | Sundries. | Total. |
|---|---|---|---|---|
| | *d.* | *d.* | *d.* | *d.* |
| Geldenhuis . . . . . . | 6·66 | 0·35 | 0·61 | 7·62 |
| Witwatersrand . . . . | 6·67 | 1·48 | 2·07 | 10·22 |

A contingent benefit arising from sorting is the opportunity it provides for removing drill-bits and other fragments of steel, iron and wood from the milling dirt, and thus reducing the risk of injury to crushing plant and of interference with the issue through the battery screens.

RESULTS OF SORTING.

| — | Waste. | Waste Value. | Cost of Sorting. | — |
|---|---|---|---|---|
| Brazil: | % | dwt. | *s.* *d.* | |
| Morro Velho | 13·12 | .. | .. | |
| St. John del Rey | 11·00 | .. | .. | |
| India: | | | | |
| Ooregum | 14·00 | .. | .. | |
| Queensland: | | | | |
| Mount Morgan | 39·89 | .. | .. | |
| Transvaal: | | | | |
| Angelo | 15·49 | .. | 1 4½ | Includes tramming and breaking. |
| Bonanza | 28·63 | .. | 1 3½ | Breaking cost, 8½*d.* in addition. |
| Crown Deep | 16·02 | 1·14 | .. | |
| Driefontein | 12·52 | .. | 1 9¾ | Includes tramming and breaking. |
| Durban Roodepoort Deep | 41·63 | .. | .. | |
| Ferreira | 35·02 | ·84 | .. | |
| French Rand | 27·35 | .. | .. | |
| Geldenhuis | .. | .. | 7¾ | Includes tramming and breaking. |
| Geldenhuis Deep | 13·57 | 1·30 | 5½ | |
| Glen Deep | 28·90 | ·74 | .. | |
| Jumpers | 39·16 | .. | 11½ | Includes tramming and breaking. |
| Jumpers Deep | 33·88 | 1·40 | .. | |
| Langlaagte Deep | 20·27 | .. | .. | |
| Meyer and Charlton | .. | .. | 5½ | |
| New Comet | 23·89 | .. | 1 2½ | Includes tramming and breaking. |
| New Goch | 20·52 | .. | .. | |
| Nourse Deep | 20·79 | 1·03 | .. | |
| Paarl | 11·49 | .. | 11¾ | Includes breaking. |
| Robinson | 33·67 | 1·00 | 6¾ | Includes first breaking. |
| Rose Deep | 19·23 | ·83 | .. | |
| Van Ryn | 22·70 | .. | .. | |
| Village | 24·50 | .. | .. | |
| Witwatersrand | .. | .. | 10¼ | Includes breaking. |

A typical picking-floor is that at the Ferreira mill, which is 70 ft. long and 11 ft. wide. Along one side, and 3 ft. above the floor, run a succession of grizzlies, 13 ft. long, and set at

40° with 1 in. spaces; above them is a track carrying side-tipping trucks, which bring ore from the shaft-head bins and distribute it along the grizzlies. The floor is laid with ½-in. steel plates, on which the ore is spread by heavy rakes, and washed by jets of water under about 18 lb. per sq. in. pressure, the waste being picked out and thrown into trucks that travel along the other side of the floor, next the clean ore bin, into which the picked ore is shovelled. The dirty water is led to settling tanks for re-use, and the settlings are periodically sent to the battery. With grizzlies set at 2¼ in. spaces, the rejections are 26·53%, at a cost of 6*d.* per ton thrown out; while at 1 in., they are 17,811 t. out of 48,540 t., or 36·69%, at a cost of 1*s.* 5·4*d.* per ton rejected, or 10·1*d.* per ton milled.

Sorting tables are made on various lines, but the main principle is the same in all. The ore is fed continuously on to a horizontally rotating plane, making one revolution in 1¼ to 1½ minutes. The picking surface is about 4 ft. wide for a single row of sorters arranged around the outer circle; it is made of ⅝-in. steel plate, and is actuated by gear beneath. A fixed plough removes the clean ore into a shoot leading to a bin, while the waste is thrown into a central hopper, from which it is periodically trucked. Or there may be a second row of pickers around the inner circle of the table, with a raised annular terrace on which the waste is piled, and from which it is continuously pushed by a fixed plough in the same way as the clean ore. At the Crown Reef mill, the sorting of 15·21% for over 50,000 t. cost 3*s.* 9·683*d.* per ton rejected, or 8·049*d.* per ton milled. These figures are greatly in excess of those recorded of picking-floors, and the work is not so well done on account of limited space. Maintenance cost is high, compared with the scale of native wages on the Rand; but in countries where labour is expensive, the table and its automatic feed and delivery may show an economy over the floor method.

Picking-belts of several kinds have been tried in S. Africa, but are there said to be costly in maintenance and of restricted capacity, their only advantage being that they may be made

to serve as conveyors, and to deliver the clean ore at a few feet higher elevation than they receive the mixed ore for sorting. Yet in the United States they are most successfully employed in the iron and zinc mines. One at the North Mine Hill, Franklin, New Jersey, in three years has carried over 350,000 t. of heavy crystalline ore in pieces about the size of 5-in. cubes. The attendants are provided with hammers, with which they break, right on the belt, the larger pieces which

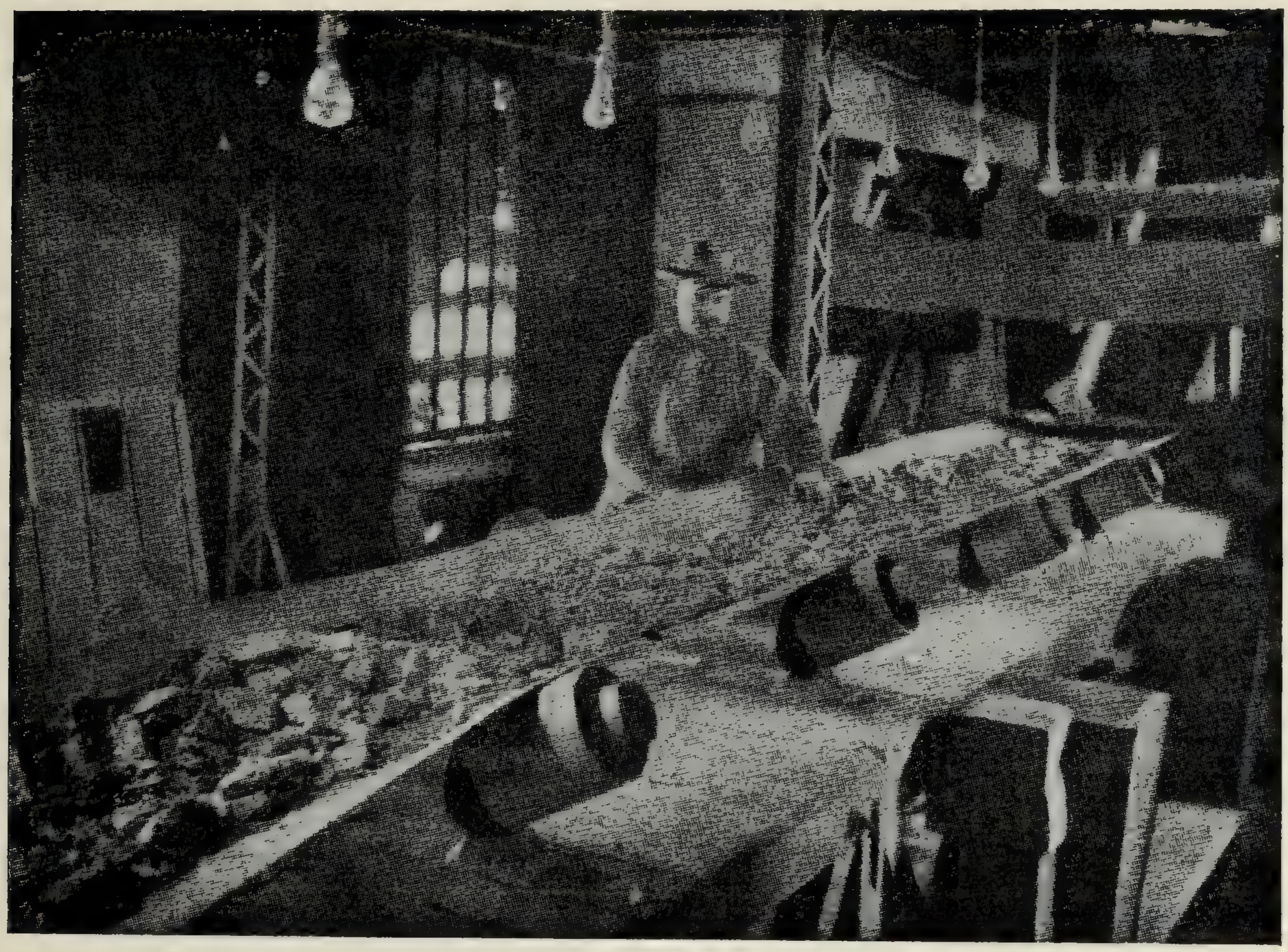


FIG. 10.—ROBINS PICKING-BELT.

have passed through the coarse crusher. The belt is made wide and heavy, and the idlers are so shaped as to give it a broad flat surface at the centre, with narrow, very slightly raised sides. The advantages claimed for sorting belts over wood or iron sorting tables are that there are no small crevices where pieces of ore can stick and jam ; no teeth or links to wear and get out of pitch ; and no bearings exposed to dust or grit. The belt will, from its elasticity, stand for years an amount of pounding and abuse that would destroy in a short

time a machine more rigidly constructed. It is made by the Robins Conveying Belt Co., of New York, and is shown in Fig. 10.

A picking-belt is used at the Meyer and Charlton, and their cost is the lowest of any Transvaal mill (5·54*d.* per ton).

## CHAPTER II.

### BREAKING.

It is hardly necessary to insist that the process of reducing rock from the size in which the mine stopes deliver it down to the size at which its gold contents can be extracted should be gradual. That is to say, it is obviously contrary to all mechanical and economical laws to attempt fine reduction in one operation. Yet this first principle in rock-crushing is very much ignored, and on many goldfields may still be seen the reprehensible practice of shovelling into a stamp battery lumps of hard quartz of the maximum size which the feed opening will admit. These lumps have to be crushed from 3 or 4 in. diam. into pulp of $\frac{1}{30}$ to $\frac{1}{40}$ in., and during the time occupied by the stamp in effecting a preliminary reduction (say to $\frac{3}{4}$ or 1 in.) it can do but little else, as the smaller particles are protected by the larger. Thus the efficiency of the stamp is very much reduced, and at the same time it is subjected to strains which entail breakages, thus adding to the cost of the operation.

**By Hand.**—It has therefore become a well-established custom to provide machines called "breakers," for the purpose of preparing the ore for reception by the fine-crushing plant. There are many small mills, however, where the initial cost of the breaker is considered an obstacle, and where, the batteries being old-fashioned and not furnished with automatic feeders, the services of a lad or man are required in any case for feeding. Usually one man serves 10 to 15 head of stamps, and such part of his time as is not occupied in the feeding operation is free for hand-breaking the largest lumps. Two evils

are manifest in this practice. In the first place, the feeding is bound to be less regular and more spasmodic than in the case of a positive automatic feeding machine; and in the second place, the human tendency to shirk work will result in the breaking being performed on the minimum scale which circumstances will allow.

*In the Mine.*—A certain amount of hand-breaking is quite legitimate, but this should be done in the mine. Varying conditions govern the size to which rock is broken in shooting down, but economical mining work calls for a minimum of explosive to detach the ore, and not so much as will have a smashing effect upon it. Hence masses as large as one's body often occur, and must be reduced before they can be loaded into trucks or skips. It is doubtful whether anything larger than a man's head should leave the mine, because of obstructions created in passes and shoots; but the bad light underground does not favour carrying the operation any further. At the Lucknow mines, New South Wales, where much of the stone is extremely hard, the author found it profitable to employ special men, wielding 20-lb. hammers, to break everything down to a maximum of 8 in. diam., to suit the mouth of the "breaker," and even then the machine refused many of the hardest pieces, especially if rounded in form. The men were selected from among the truckers (who received 5*s.* 6*d.* a day), and were paid an extra 6*d.* to 1*s.*

*On Surface.*—When surface hand-breaking is resorted to, the first consideration is a solid and firm floor and a sufficiency of light, and the situation should be chosen with a due regard to saving labour in feeding the stamps with the broken ore. Sometimes the floor is covered with iron, such as old flat sheets from the mine, and this facilitates shovelling, but it is very difficult in practice to bed them so evenly as to entirely avoid spring. Nothing forms a better foundation than broken stone, and if a good layer of this be provided there will be little risk of losing the finest material, which is always the richest. A 3 to 5-lb. hammer should suffice.

*Costliness of.*—But on no ground can hand-breaking and feeding be justified if the mine be worthy of any crushing

plant at all. The additional cost of a breaker and self-feeders is repaid in a very short time by the increased duty of the stamps and lessened cost all round. It is not often that a direct example can be quoted in proof of this, but it happened to the author in 1895 that he was controlling two adjoining properties, working on precisely the same formation, and milling ore that was practically identical. The two batteries were situated (at Lucknow) in New South Wales, where, as in Victoria, breakers are unfashionable. The subjoined figures speak for themselves, it being premised that the Wentworth mill used a breaker and self-feeders while the Aladdin mill had neither :—

MILLING, COST PER TON.

| — | Short Tons Milled. | Duty per Stamp. | Wages. | Fuel. | Stores. | Maintenance. | Total. |
|---|---|---|---|---|---|---|---|
| | | tons. | *s.* *d* | *s.* *d.* | *d.* | *d.* | *s.* *d.* |
| Wentworth | 11344 | 1·869 | 1 7·64 | 0 6·61 | 4·36 | 11·90 | 3 6·51 |
| Aladdin . | 4768 | 1·028 | 3 7·19 | 1 6·13 | 10·46 | 7·61 | 6 7·39 |

The contrast is sufficiently striking, though the figures do not fully express it. The items fuel, stores, and maintenance should be directly proportionate to work done, while labour is not so of necessity, because the men may be either overworked or underworked. As a matter of fact, in the Aladdin mill they were kept very fully occupied, while in the Wentworth they were comparatively slack, which is proved by the circumstance that in 1898 the same staff of men managed 35 stamps instead of 20, milling 14,481 t., at a total cost of 1*s.* 7·47*d.* per ton, notwithstanding that, owing to increased hardness of the stone, the duty per stamp was reduced to 1·54 t. Moreover the Aladdin mill had a slight advantage, in dealing with richer and a trifle softer rock. The difference between the two costs, amounting to say 3*s.* a ton, represents over 700*l.* on one year's work—a sum vastly more than sufficient to provide a breaker and self-feeders.

It must, however, be admitted that this example is taken from a country where labour is expensive, mill-hands receiving 9*s*. per shift of 8 hours. On the other hand, fuel was not cheap, wood costing 9*s*. a cord, and inferior coal about 14*s*. 4*d*. per ton. But it is possible to imagine a combination of circumstances—such as limited and costly power, lack of skilled labour and very low wages for unskilled labour, where it would be injudicious to introduce breakers. Such must, in any case, be so rare as to serve only to emphasise the rule.

**By Machine.**—The work done by a rock-breaking machine is most violent and spasmodic, from which it follows that the strains and stresses are enormous, and that considerable vibration must be generated. On these grounds, and with due

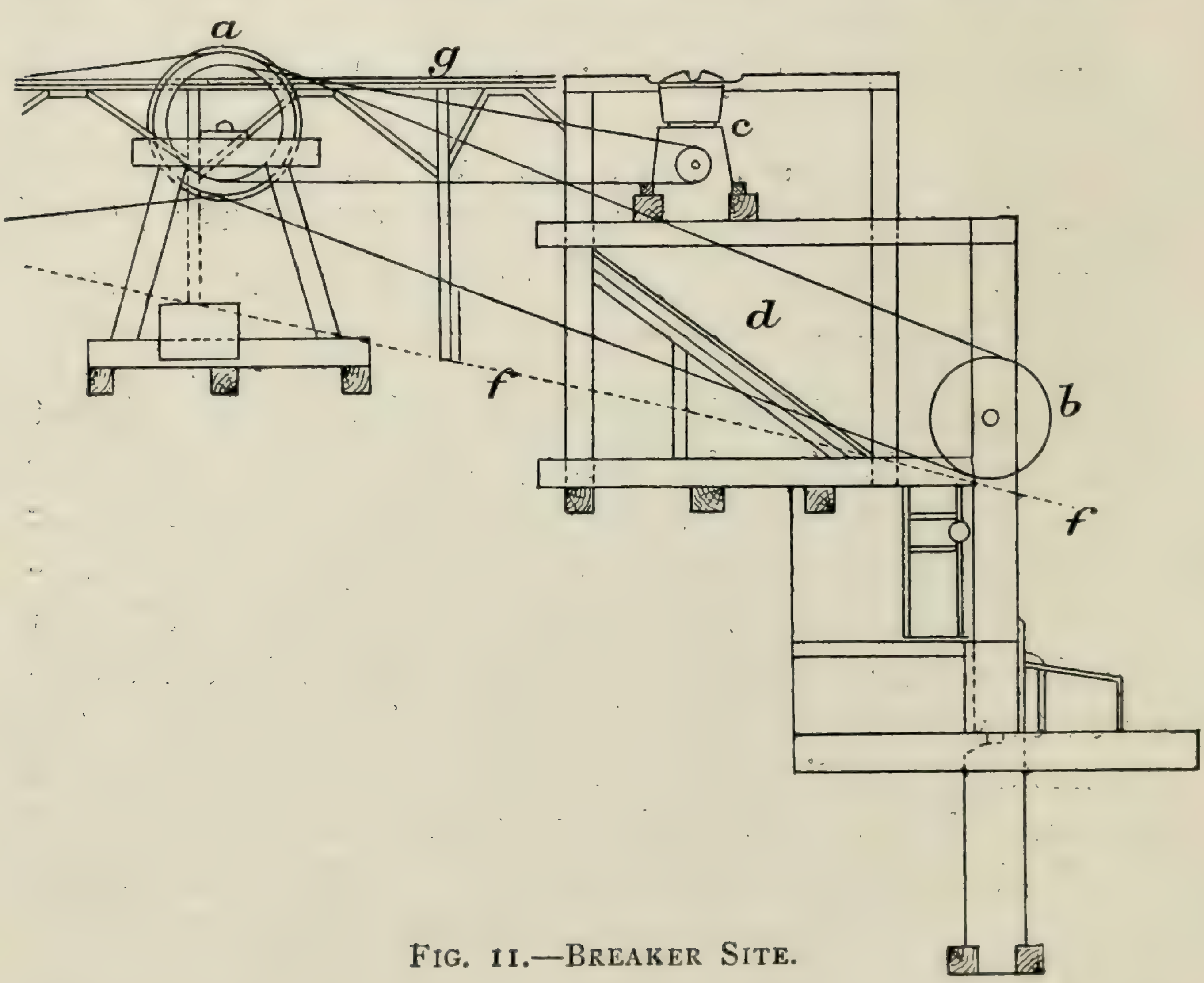



FIG. 11.—BREAKER SITE.

consideration for facilities of feeding the breaker from bins, grizzlies and sorting platforms, and of delivering its product to the next stage of crushing, the situation demands very careful attention.

*Site.*—The foundations of such ponderous machines, weighing as they do from $1\frac{1}{2}$ up to over 40 t., is a matter of primary importance, and accessibility for renewal of wearing parts is

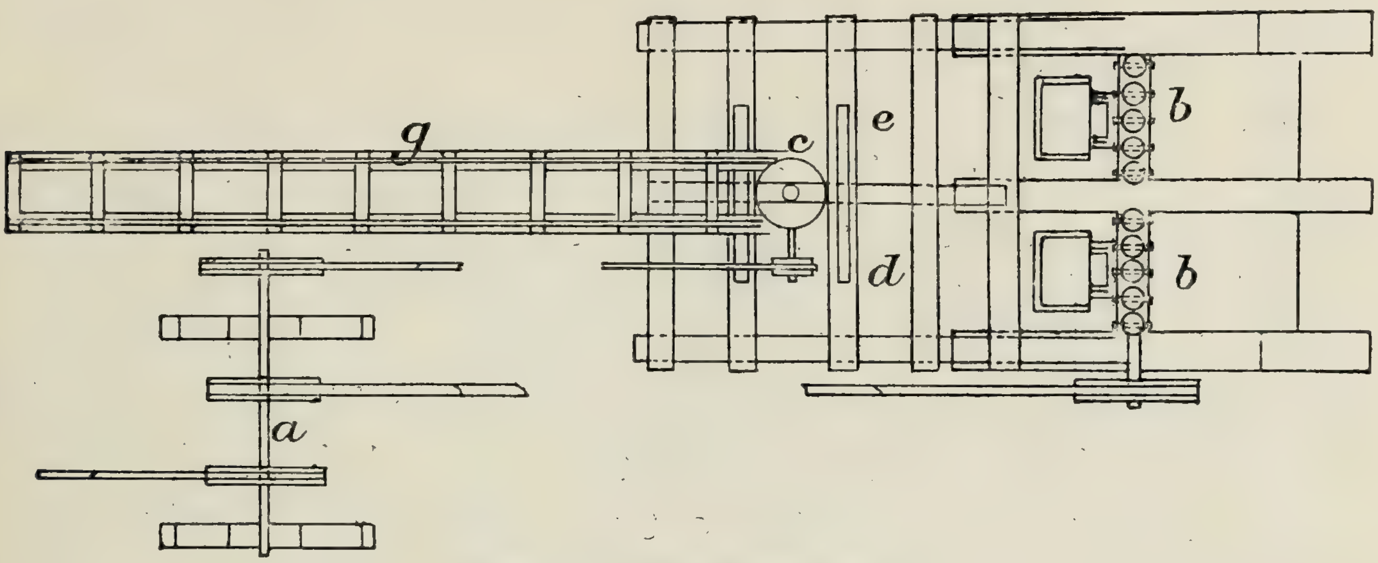



FIG. 12.—BREAKER SITE.

of almost equal moment. It used to be a stereotyped practice to erect the breaker above the battery bins. This may be excusable in the case of a small plant, where the same motor and counter-shaft have to do duty for both breaker and battery, and where the structure is placed on a naturally sloping surface, so that the machine stands at no great height above its foundations. Such an example is illustrated in Figs. 11 and 12, where the same shafting *a* drives both the 10-head battery *b* and the gyratory breaker *c*, the issue from the latter being easily directed into either of the bins *d e*. As will be seen from the dotted line *f*, indicating the contour of the ground, the construction is exceptionally solid, the lower part of the bins *d e* and the whole of the battery being on natural bedrock. Moreover, the location of the breaker is restricted in this case by the necessity for making provision to receive (by the same tramway *g*) public ore for custom milling as well as the product of the mine alongside. In this battery also there are no vanners. The driving pulleys have 24 ft. centres between engine and countershaft, 19 ft. between countershaft and breaker, and 33 ft. between countershaft and battery.

While the arrangement just described was adopted by the author to suit local conditions, he had occasion to condemn a

somewhat similar one where the circumstances did not justify it, in the Wentworth battery at Lucknow (N.S.W.). Here a swing-jaw breaker, run from the battery engine, was placed at the extreme top of the building, resting upon the framework of the battery bins, as indeed is very commonly the case. The results were objectionable on every score. Effecting renewals of wearing parts was rendered very awkward, and therefore the task was shirked as much as possible, while the inaccessibility of the machine made supervision difficult. The violent oscillations inseparable from this type of breaker, continued for a length of time, and, coupled with the thrust of the loaded bins and the pull of the driving belts, had forced the battery posts completely out of plumb, lessening the efficiency of and augmenting the repairs to the battery. Moreover the load on the engine was at times very severe, and it was impossible to secure an even and regular speed for the stamps; and the dust emanating from the breaker was productive of endless trouble with the bearings throughout the mill, and especially those of the more delicate vanners which are under the same roof.

Whenever the scale of operations is at all considerable, it is well worth while to let the breaker or breakers be kept quite separate from the battery. In this way, choice can be made of the best location, having in view the various requirements of economic running. A "crusher station" is an accepted feature of the great Transvaal mills, and is not unknown in America and Australia.

Sometimes, as was done by the author at the Wentworth mill, a ground site is chosen. In this instance, a hoist carrying ½-ton trucks, operated by the battery engine, was in use for raising ore as it came from the adjacent shaft to the breaker at the top of the battery. A breaker station and bins were adapted to this existing arrangement, at the same time accommodating ore to be delivered by surface tramway from two other shafts which previously had not been used for the purpose. Some rock excavation was necessary to convey the broken stone to the elevator, but a compensating advantage was that the breaker bins were in solid ground, and that

the machines themselves were set on beams laid on bedrock. Fig. 13, from a photograph, shows the relative positions of bins, breakers and motive power, the countershaft being outside the right of the picture. With plenty of light, and ease of access for renewals, the feed platform being only 4½ feet above the roadway, all work was very much better done.

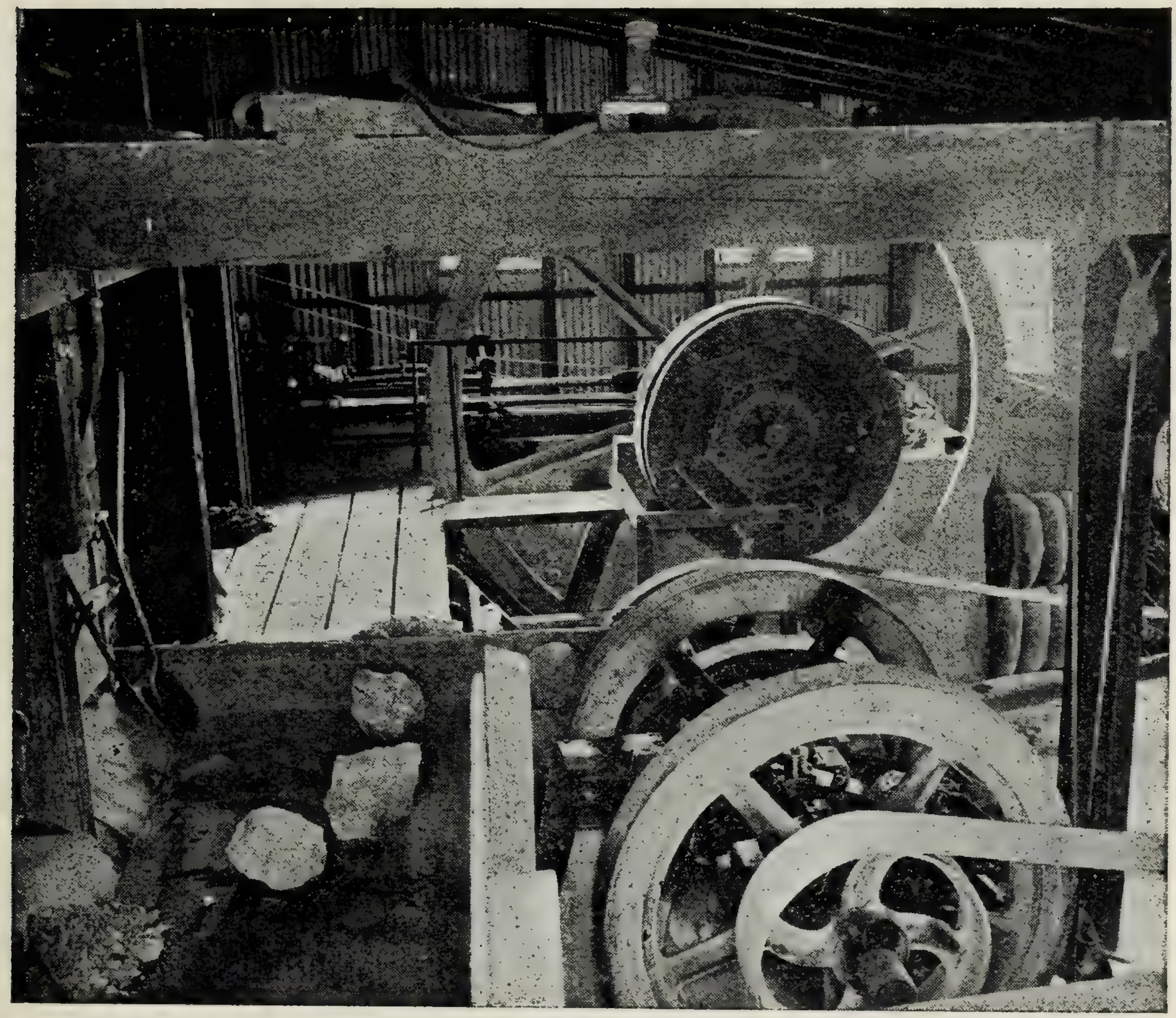


FIG. 13.—BREAKERS.

Another advantage secured was that the battery engine, which had previously only sufficed to drive 20 stamps, and could barely maintain 85 drops a minute while the breaker was running, easily managed 35 stamps at 98 drops when the breaker was removed.

Where operations are on a very large scale, it is convenient, and often done in S. Africa, to place the grizzlies and breakers along with the sorting table on the shaft-head

platform. This implies a very extensive arrangement, because a proper installation should provide bins at every stage, so that each operation may proceed regularly without

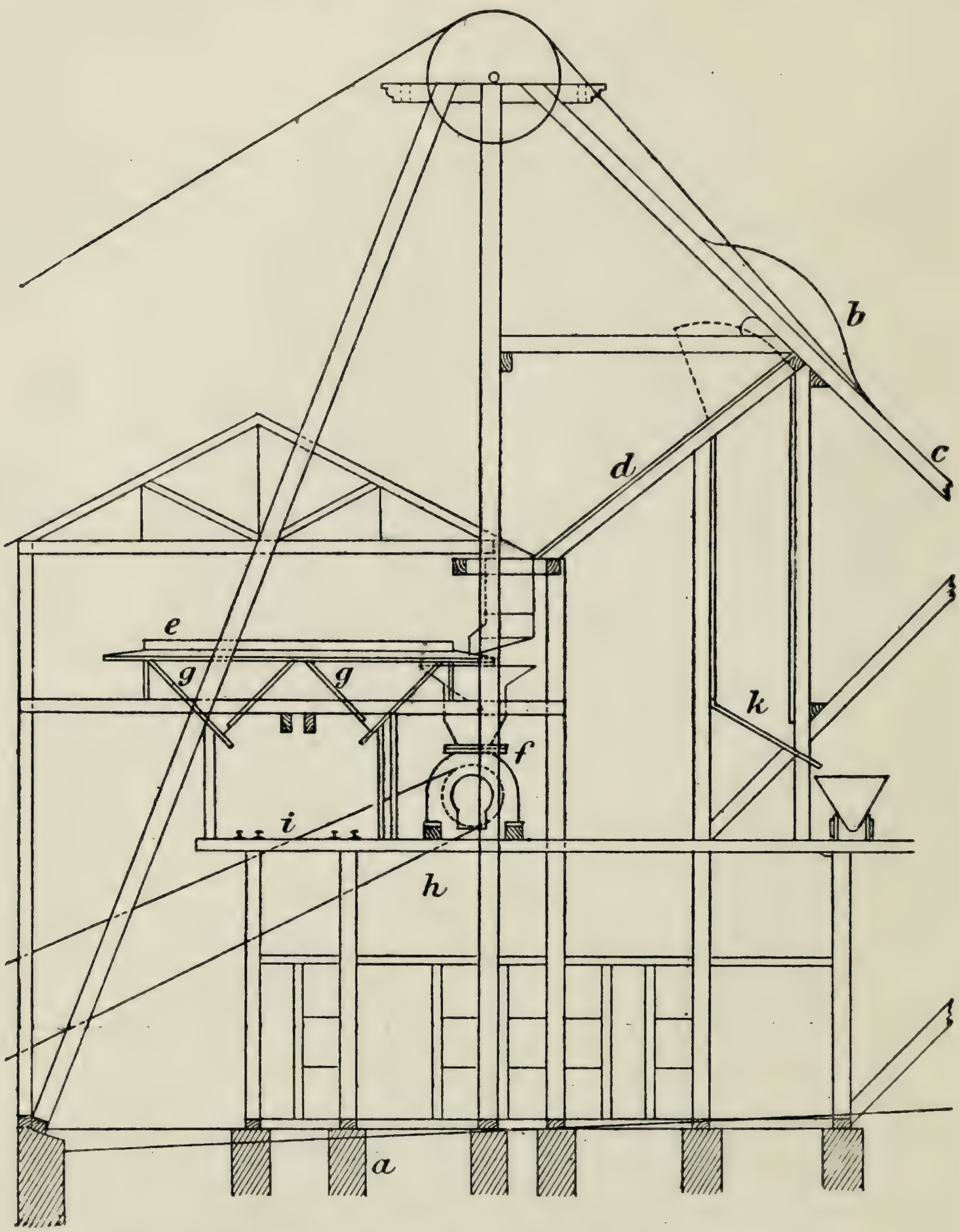



FIG. 14.—SORTING AND BREAKING ON SHAFT-HEAD.

being directly controlled by the one preceding or following it, and few head-gears are of such magnitude as to encompass it. Fig. 14 (scale $\frac{1}{16}$ in. = 1 ft.), adapted from Truscott's work, shows an example where the shaft-head structure *a* is availed of; *b* is an automatic tip for trucks arriving by incline *c*; *d* is

a grizzly delivering to picking-table *e*, the clean ore from which is ploughed off directly into gyratory breaker *f*, and falls into bins *h*; while the rejected waste, thrown into hoppers *g*, passes to trucks at *i*, and (with mullock sent up from below and tipped into *k*) is trammed away to the dump.

*Motive Power.*—In few machines is the strain greater and more erratic than in a breaker, especially in the swing-jaw pattern. The amount of feed, the hardness of rock, and more particularly the size of the lumps, is subject to continual and most sudden variation. The load on the motor is never constant, and fluctuates between very wide extremes. It is therefore most desirable that the power for driving breakers of any but quite small size should be derived from a source independent of all other machinery. The machine itself should have very heavy fly-wheels, and the engine driving it should be similarly provided, and be equipped with a very sensitive governor. As breakers run at great speed, and are liable to sudden fracture under excessive strain, it is well to avoid putting anything of value immediately in front of the fly-wheels, in case of accidents; they have been known to fly off and create wholesale wreckage. As to the amount of power required, it is well to make a very liberal allowance beyond that stated by the makers. No doubt they quote a fair average for moderately hard rock; but an average figure is no guide to the actual needs of this class of machine, and what is absolutely necessary is a margin beyond the extreme maximum—that maximum being a surprisingly high unit when an extra large lump of unusual hardness or toughness is suddenly encountered.

*Dust.*—Unless the ore delivered to the breaker is in a very wet condition, the crushing operation engenders a vast amount of dust. This represents a direct loss in value, experience showing that the finest particles always carry the highest proportion of gold, due probably to the fact that the mineralised and auriferous portions of the stone are more brittle and pulverulent than the grains of barren quartz. Further, the dust is a great inconvenience to the men, and highly destructive to machinery; bearings suffer very much, causing loss of power by increased friction, waste of lubricant,

and wear and tear. The dust nuisance forms an additional argument in favour of placing the breaker in a separate building. It can, however, be moderated to a considerable extent by applying a spray of water. This was done at Lucknow, the tap regulating the sprinklers being placed within easy reach of the hand of the man in charge. But as the crushing proceeds, further dust is created, out of reach of any shower, and this it seems impossible to prevent. Excess of water must, of course, be avoided, or the fine stuff will pack in the shoots.

*Types.*—There are two types of rock-breaking machine. In the older form, the action consists in repeatedly forcing one flat face towards another, the ore being interposed between the faces; this is known as the swing-jaw or reciprocating principle. The newer form has a conical crushing head gyrating within a conical rim at a distance that admits the ore; breakers on these lines are called gyratory.

*Swing-jaw Breakers.*—The original pattern was made by W. P. Blake, of New Haven, Connecticut, U.S., and his machine is still in great request, especially in the United States. He was followed by numerous imitators, who introduced modifications aiming at, but not always successful in effecting, improvements. To mention a few, there is the Dodge, an American machine; the Blake-Marsden, by Marsden of Leeds; the Hope and the Jaques, of Australian manufacture; and so on.

The Blake machine, as made by Fraser & Chalmers, is shown in Fig. 15. *a* is a very massive frame, cast in one piece, receiving and supporting all the other parts; it has feet *b*, provided with bolt holes, so that it may be fastened down firmly on concrete or timber foundation. Care must be taken that it is set absolutely level, and with the driving shaft accurately lined. The driving pulley *c*, which will run either way, should make 225 to 250 rev. per min.; *d* are heavy fly-wheels. The pitman *e* connects the eccentric *f* with toggles *g*, and the motion thus transmitted causes swing-jaw *h* to approach and recede from fixed-jaw *i* at each revolution of the shaft. The swing-jaw *h* is pivoted freely on the iron bar *k*, and the fixed-jaw *i* is bedded in zinc ¼ in. thick against the

frame *a*. Each jaw is faced with a corrugated wearing plate *l*, made reversible, so that when the bottom edge is worn away it can be turned upside down; that in *h* rests in a ledge,

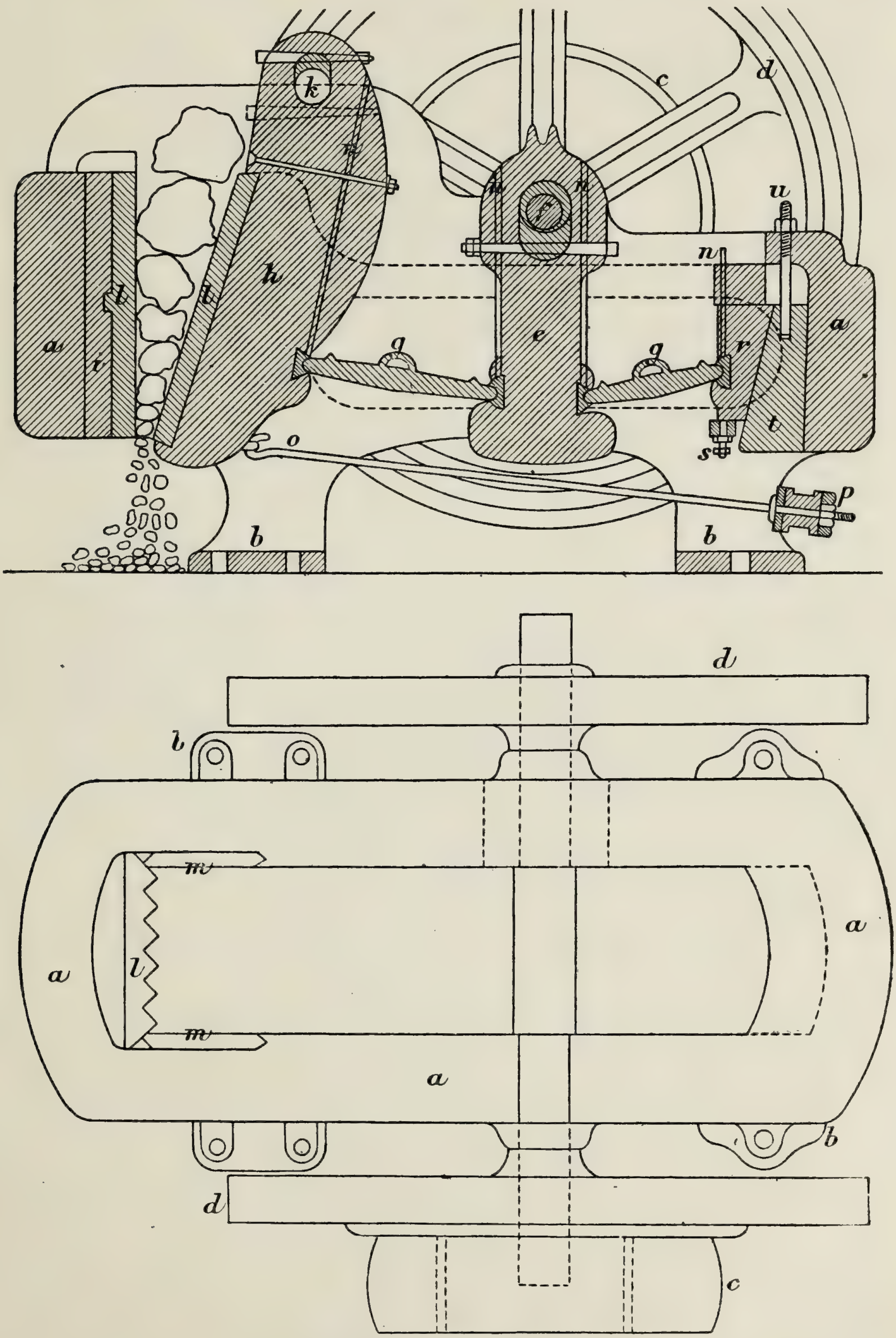



FIG. 15.—BLAKE BREAKER.

while that in *i* is kept in place by cheek pieces *m* fitting into recesses in the frame. Lubrication is effected by oil tubes *n*. The tension rod *o* and rubber spring *p* help to pull the swing-jaw back after each stroke. The toggle block *r* is tightened by means of set-screw *s*; and the size of stone broken is regulated by raising or lowering wedge *t* by aid of screw *u*.

When setting up, having fixed the frame level and true, put in swing-jaw *h*, tighten caps on its shaft *k* sufficient to keep it from moving, and then put on lock nuts. Next insert pitman *e*, with large end of key nearest toggle block *r*; let it drop on to a wooden block high enough to clear the bearings about 6 in., then slide in the shaft, taking care to have the right end for the driving pulley. Put in the lower box with thin wood packing, which will keep the key from tightening the lower box to the shaft; insert key from back, and tighten set-screw. Lower shaft into bearings and put on caps, having a strip of thin wood or leather under the caps to prevent them from being screwed down too tight on the shafts. Hang fly-wheels *d* according to marks on shaft, key them tight to clear bearings by $\frac{1}{16}$ in, and screw driving pulley *c* in place. Insert toggles *g*, longest in front or between swing-jaw *h* and pitman *e*. Screw wedge *t* down to lowest point, and add rod *o* and spring *p*, not screwing tighter than necessary. Tighten toggle block *r* with set-screw. Oil by way of tubes, and take care to keep them plugged to exclude dust.

The subjoined table gives capacities, dimensions, etc., of some standard sizes:—

BLAKE BREAKER.

| No. | Mouth. | Weight. | Output. | Length. | Width. | Height. | Pulley diam. | Face. | Height of feed. | Fly-wheel diam. | Power required. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | in. | t. | t. | ft. in. | ft. in. | ft. in. | in. | in. | ft. in. | in. | h.-p. |
| A | 10 × 4 | 2¼ | 4 | 4 6 | 4 0 | 3 11 | 20 | 6 | 1 10¾ | 42 | 4 |
| 2 | 10 × 7 | 4 | 6½ | 5 4 | 4 6 | 4 5 | 24 | 8½ | 2 8 | 44 | 7 |
| 5 | 15 × 9 | 8 | 11½ | 6 8 | 5 0 | 5 3 | 30 | 11 | 3 1¼ | 58 | 10 |
| 8 | 20 × 10 | 8½ | 13 | 6 10 | 5 9 | 5 11 | 36 | 12 | .. | .. | .. |

These figures are given on the authority of the makers The output is calculated on a 2-in. product, i.e. when the jaws are set at $1\frac{1}{2}$ in. open at the bottom ; the rate is per hour, assuming full speed and constant feed. Such results, with the power named, cannot be counted on in practice. From the author's own experience of this type of machine, giving a product not exceeding $1\frac{1}{2}$ in., he would allow at least double the average power quoted above, and would advise a good margin beyond that even for occasional extra heavy work on unusually hard stone.

The Blake-Marsden machine is shown in Fig. 16. It is made in about a dozen sizes, ranging from a machine with

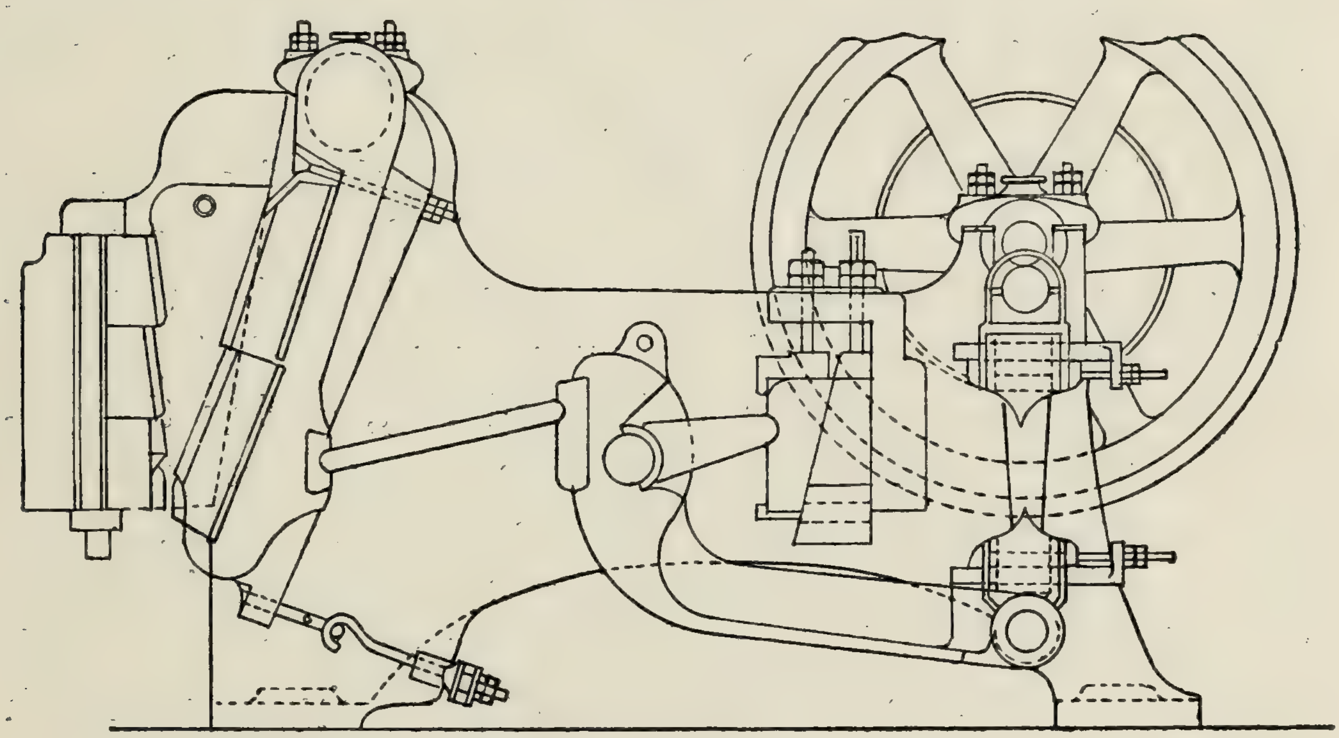


FIG. 16.—BLAKE-MARSDEN BREAKER.

jaws 12 × 8 in., weighing about 5 t., costing 113*l.*, and requiring 3 h.p. (nom.) to reduce 108 cwt. per hour to road-metal size, up to a machine with 24 × 19 in. jaws, weighing 20 t., costing 300*l.*, and needing 16 h.p. to crush 350 cwt. per hour. These figures are quoted from the catalogue issued by the maker, no others being available.

Machines of this group demand an exceedingly firm seating, on account of the effect of the incessant oscillation of the heavy swing-jaw. The wear and tear is very much concentrated upon the lower edges of the two jaws, so that they become fretted and chipped away while the main portion of the face plates remains almost as good as new. The

tendency too is for the feed to seek the corners, whereby the cheek plates are rapidly worn, and their constant renewal is an essential for keeping down the gauge of the product. In some adaptations of the Blake principle, the tension rod for drawing back the swing-jaw is a constant source of trouble, especially where steel springs are used instead of the rubber cushion. Nevertheless the machine is eminently powerful and simple, and is well adapted for breaking big stone down to 1½ to 2 in.; even where a finer product is desired, it is often retained as a first crusher.

The great weight of machines made on the Blake pattern, owing to the massive cast-iron frame, is a decided drawback when cost and difficulty of transportation have to be considered, and it has led to attempts at reducing the weight without diminishing the strength.

A Canadian breaker, called the "invincible," endeavours to lessen the strain, and thus admit of a lightened frame, by giving the jaw a nearly parallel motion and making the outside one movable. The jaw moves over an elliptical path, the long axis of which is so inclined that, during the moment of crushing, the swing-jaw plate moves directly toward the stationary plate, and, at the end of the stroke, moves down and back to let the material pass through, tension being taken by four wrought-iron rods running through to the back of the frame. The machine seems to closely copy a much earlier American prototype which has quite fallen into disuse.

On the other hand, there is great practical utility in a modification of material instead of in structure introduced by Hadfields, Sheffield. By adopting the exceedingly tough manganese steel made by this firm in lieu of cast-iron for the main frame and for the jaw plates, the machine can be made of half the weight with three times the strength; and while the cost of the various sizes is practically only the same as for the corresponding sizes in cast-iron, the lessened weight means considerable economy in freight and transport. Moreover the manganese steel frame is virtually unbreakable, which certainly cannot be said of the cast-iron.

In the Dodge breaker, the swing-jaw is pivoted below

instead of above, and for this reason much greater uniformity in size of product can be secured; moreover it is possible to set the jaws much more closely together, and to thus obtain a finer product if desired. But all these gains are at the expense of capacity, and the practically fixed relative positions of the two jaws at the extreme point of exit permits a certain amount of wedging of pieces of stone just too large to escape. The machine is in no sense a competitor with the Blake principle, as its capabilities are far inferior; but it is sometimes used as a supplementary adjunct, to reduce the Blake product to a somewhat finer gauge. It can be very easily adjusted. A few figures concerning it, taken from the maker's catalogue, are appended :—

DODGE BREAKER.

| No. | Feed. | Output per hour. | Weight. | Pulley diam. | Face. | Speed per min. | Power required. |
|---|---|---|---|---|---|---|---|
| | in. | t. | lb. | in. | in. | rev. | h.-p. |
| 1 | 6 × 4 | ½ to 1 | 1200 | 20 | 4 | 350 | 2 to 4 |
| 2 | 9 × 7 | 1 to 3 | 4300 | 24 | 5 | 300 | 4 to 6 |
| 3 | 12 × 8 | 3 to 4 | 5600 | 30 | 6 | 250 | 7 to 8 |
| 4 | 15 × 11 | 8 to 10 | 13500 | 32 | 12 | 200 | 12 to 18 |

A recently introduced breaker, called the "roll-jaw," is of jaw and toggle type, but the movable jaw is given a rolling motion, which is intended to render the crushing finer and more uniform, and is claimed to absorb less power and to exert less wear on the working faces. Doubtless the work is more distributed, and it is quite conceivable that a rolling action may be more effective on rock which is not too hard; but it also seems likely that the rounded face of the jaw will push really hard rock in front of it, and fail to grip as a flat jaw would. The assertion that it will "take in hard rock and reduce it at once to gravel or sand" is useless without some evidence as to quantity per hour and hardness of material. That a single machine should be capable of doing the work of coarse breaker, fine breaker, and rolls, in one operation,

and that economically, will be doubted by most engineers, even though it be turned out by such a well known firm as the Sturtevant Co.

*Gyratory Breakers.*—Typical of and prominent among these is the well-known breaker bearing the name of the

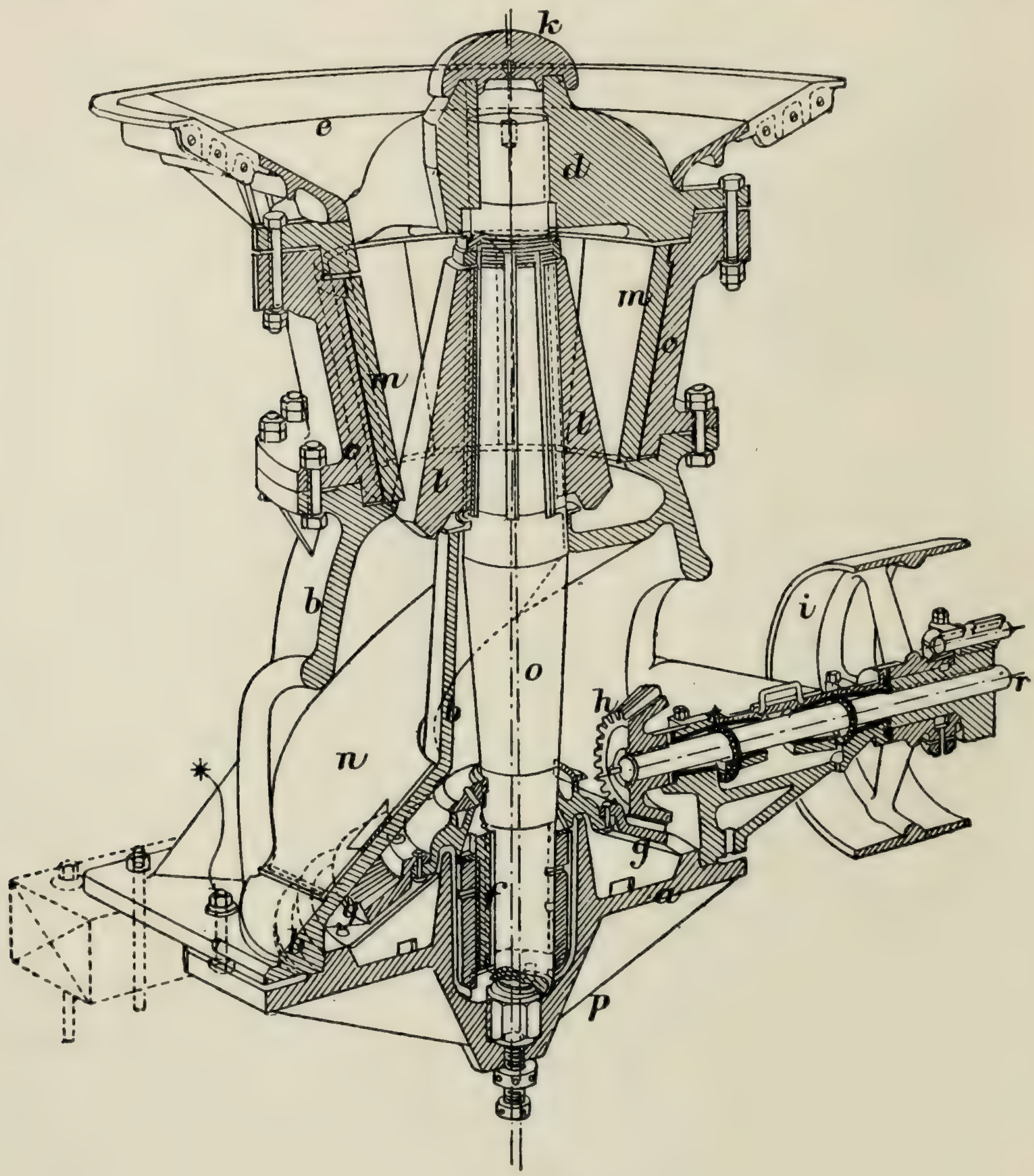



FIG. 17.—GATES BREAKER.

Gates Iron Works (Chicago, Ill., U.S.A.), shown in Fig. 17: *a*, bottom plate; *b*, bottom shell; *c*, top shell; *d*, spider; *e*, hopper; *f*, eccentric; *g*, bevel wheel; *h*, bevel pinion; *i*, driving pulley; *k*, dust cap; *l*, crushing head; *m*, concave dies; *n*, wearing plates; *o*, main shaft; *p*, steel foot-step; *r*, counter shaft. The ore to be crushed is fed in at *e*, the spider

*d* dividing the space into two or more compartments. The vertical shaft *o*, of forged steel, is given a gyratory motion by means of the eccentric box attached to the bevel wheel *g*, which is set $\frac{1}{2}$ in. out of centre ; and thus the crushing head *l*, borne on it, in conjunction with the fixed concave dies *m*, breaks up the rock in its downward passage. The gauge of the product is controlled by regulating the distance apart of the dies *l m* at their lowest point, by means of set-screws, and a much finer result is possible than with a swing-jaw breaker. It will, however, be obvious that this is a much more complicated machine, and that while it is tolerant of a notably heavier feed, and possesses a higher capacity both in quantity and fineness, it has many more wearing parts to get out of order, and less accessibility for repairs. The need for moderating this last drawback has been recognised by the makers, who have recently introduced a new pattern, which they call " Style D."

The main point of difference lies in the construction of the bottom shell, and the absence of the oak frame which has hitherto formed the base of this machine, now dispensed with by extending the bottom flange of the bottom shell wide enough to form the base. When placed on a timber or masonry foundation, the construction admits of dropping the bottom plate, together with the eccentric box and gear wheel. The advantage of this arrangement is that the operator is enabled to get at the working part of the breaker without taking up the entire machine. Sufficient space is provided in the foundations to allow the bottom plate to be dropped on long bolts, exposing the eccentric box and gear. The bottom plate with the box can then be moved out on light rails or other means from under the crusher, to a point where it can be inspected and repaired. In dispensing with the wooden base frame, the makers also obviate the danger of the outer bearing of the countershaft getting out of line. On the bottom shell is cast the main bearing for the countershaft, of ample size and length. The main driving pulley is overhung, as a rule, and the old safety device or break-pin arrangement is retained. By keeping this bearing an integral part of the

shell, it must always be in alignment. With each machine is sent a mandrel for re-babbiting this box, thus putting means of repair in the hands of the operator. The countershaft is in most cases extended beyond the pulley, so that the operator may attach gearing if desired for running elevator and screen, in which case an outboard bearing may be put on.

In the annexed table are given some figures for comparison with swing-jaw machines. The difference of feed capacity in favour of gyratory machines is very marked, as also are the weight of the machine (economising in transport) and the greatly lessened vibration caused by it in work. On the other hand, where head room is of moment, the Blake breaker can be fixed where the Gates cannot, the comparisons lying between "height of feed" in the one and "height above frame" in the other. The Blake is also much lower priced. The output and power of the gyratory machine is based on a 2½-in. (instead of 2-in.) product. On the latter point, F. White* says that two Gates breakers, Nos. 4 and 2, working on Rand banket, feeding from one into the other, and reducing to pass 1½-in. screen at the rate of 20 t. per hour, at the Luipaard's Vlei mill, required 26 i.h.p. on a 72 days' run (22,533 t.).

GYRATORY BREAKERS.

| No. | Mouths (2). | Weight. | Output. | Hopper diam. | Height above frame. | Total height. | Pulley diam. | Face. | Speed per min. | Power required. | Approximate cost. | Minimum size product. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | in. | t. | t. | ft. in. | ft. in. | ft. in. | in. | in. | rev. | h.-p. | £ | in. |
| 0 | 15 × 4 | 1½ | 2 to 4 | 3 5 | 3 4½ | 7 4½ | 16 | 6 | 500 | 5 | 90 | ⅝ |
| 1 | 18 × 5 | 2¾ | 4 to 8 | 3 7½ | 3 9 | 8 0 | 20 | 7 | 475 | 10 | 130 | ¾ |
| 2 | 21 × 6 | 4 | 6 to 12 | 4 1¼ | 4 2¼ | 8 11¼ | 24 | 8 | 450 | 15 | 170 | 1 |
| 3 | 22 × 7 | 7 | 10 to 20 | 4 11½ | 5 2 | 10 2 | 28 | 10 | 425 | 25 | 250 | 1¼ |
| 4 | 27 × 8 | 10½ | 15 to 30 | 5 9 | 6 5¼ | 12 2¼ | 32 | 12 | 400 | 30 | 400 | 1½ |
| 5 | 30 × 10 | 14½ | 25 to 40 | 6 6 | 7 2¾ | 13 2¾ | 36 | 14 | 375 | 40 | 550 | 1¾ |
| 6 | 36 × 11 | 20 | 30 to 60 | 7 4 | 8 3¾ | 15 0¾ | 40 | 16 | 350 | 60 | 750 | 1⅞ |
| 7½ | 45 × 14 | 30½ | 75 to 125 | 7 8½ | 9 8 | 16 8 | 44 | 18 | 350 | 125 | 1100 | 2½ |
| 8 | 63 × 18 | 45 | 125 to 200 | 9 3 | 10 11½ | 19 11½ | 48 | 20 | 350 | 150 | 1500 | 3 |

* Trans. Inst. M. & M., vii. 129 (1898).

The adjustable Comet crusher turned out by Fraser & Chalmers is another prominent example of the gyratory class.

Although mills of the gyratory type undoubtedly give the best crushing results, it has been found that owing to the time occupied in removing the worn-out crusher head and concaves and fixing new ones, the rapidity with which the cast-iron heads and concaves wear away, together with the friction at the lower end of the gyratory shaft, and the rapid wear of the eccentric bearing, many users of this type of machine have contemplated replacing it by the less efficient but simpler jaw type. To remedy the defects mentioned, Hadfields of Sheffield have recently introduced a machine in which the frame is constructed wholly of their well-known toughened (manganese) cast steel. The gyrating shaft is hollow, and suspended internally at its upper end by a ball bearing, upon which it gyrates with practically no friction. The gyratory motion is transmitted from the eccentric to the interior of the hollow shaft, friction rollers being interposed, which greatly reduces the power required to drive the machine. The working parts also are quite inaccessible to dust, and are automatically supplied with a constant flow of clean oil.

The machine is shown in Fig. 18. The crushing head *a* is carried by a hollow shaft *d*, supported at its upper end by an upright pillar *e*, within the hollow shaft. This, which may advantageously be cast in mild steel, is closed at its upper end, and is there provided with an internal bearing *f* resting upon the top of the upright pillar *e*, whose lower end is fitted in a socket formed in a base-plate bolted to the main frame of the machine. The upright pillar *e* carries the weight of the shaft *d* and crushing head *a*, and takes the downward thrust due to the crushing operation; the skeleton cover or spider *c* laterally supports the top of the shaft *d*.

About the lower portion of the pillar *e*, resting in a box or support secured to the underside of the base-plate, is mounted a bevel-wheel *g* having a boss *h* which is eccentric to the axis about which the wheel rotates, and is formed with parallel

sides that are slightly inclined to the vertical; the eccentric boss *h* extends upward within the hollow shaft *d*, and its rotation causes the lower end of the shaft to travel in a circular path, thus imparting the necessary gyratory motion to the shaft and its crushing head. Between the eccentric

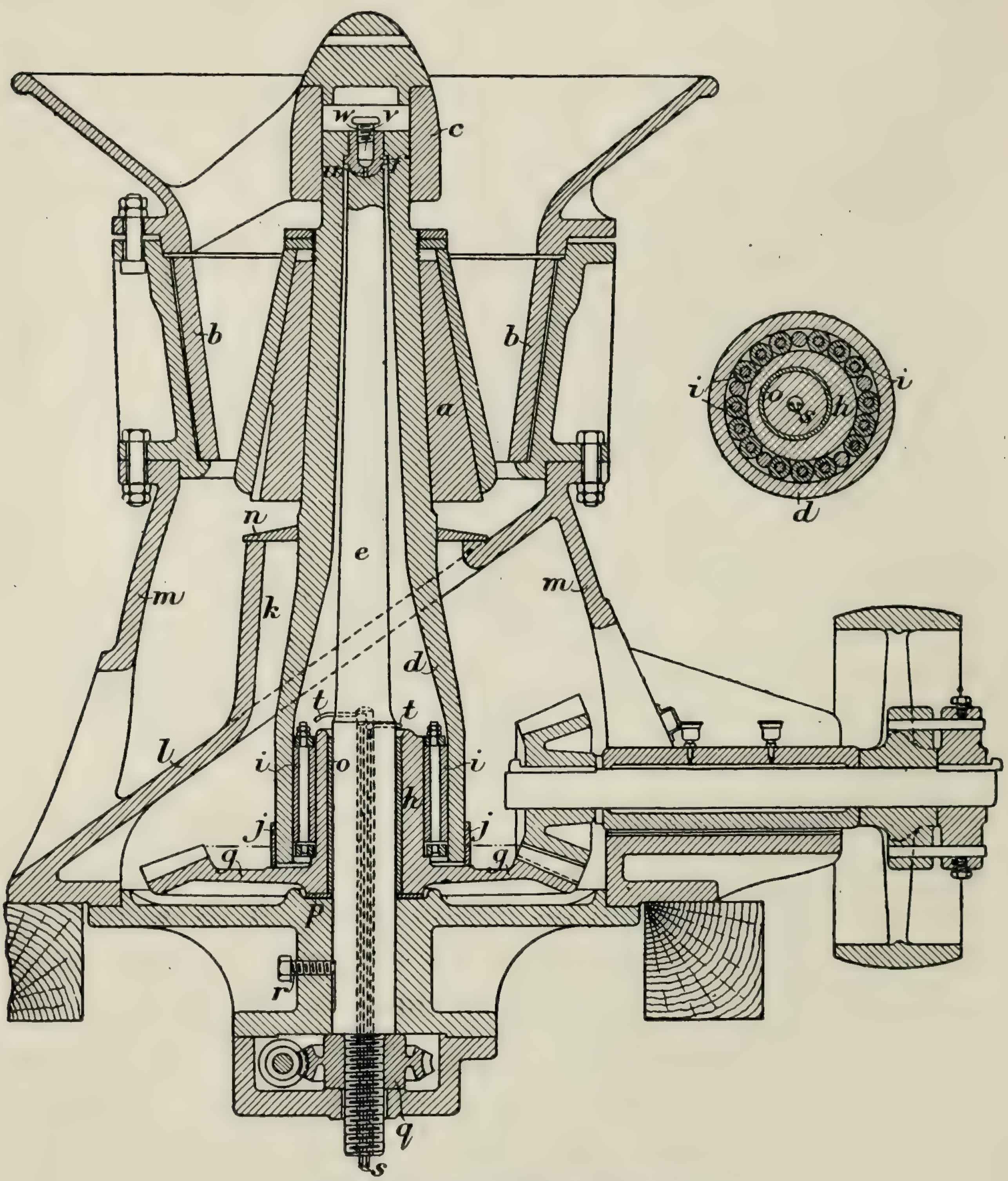



FIG. 18.—HADFIELD BREAKER.

boss *h* and the internal surface of the hollow shaft *d* is interposed a ring of anti-friction rollers *i*, carried in a frame, the lower part of which rests freely upon a shoulder formed around the eccentric boss.

To prevent dust getting to the bearing surfaces within

the hollow shaft, the upper surface of the bevel-wheel *g* is faced, and a ring *j* of metal or leather is so arranged as to loosely fit the lower end of the hollow shaft and to rest upon the faced portion of the wheel. The hollow shaft extends through a central flanged aperture *k* in an inclined diaphragm *l*, which, as usual, divides the tubular frame *m* of the apparatus, the said aperture being closed by a ring *n* fitting the hollow shaft and resting on the flange of the aperture. The upper part of the frame forms the crushing chamber and the diaphragm serves as a discharge shoot, whilst the lower part of the frame contains the bevel-wheel *g* which is driven by means of a bevel pinion from a belt pulley, as usual. Between the bevel-wheel *g* and the upright pillar *e*, a gun-metal bush *o* is provided, and a gun-metal ring *p* may also be introduced between the base-plate and the bevel-wheel *g*.

To enable the pillar *e* and consequently the crushing head *a* to be adjusted vertically, the lower end of the pillar is screw-threaded and extended through the base-plate, where it is fitted with a nut *q* supported in a bracket attached to the base-plate; the pillar is prevented from turning within its socket by means of a screw *r* projecting into a key-way, so that by rotating the nut *q*, the pillar is raised or lowered as required. By this means, wear of the crushing head *a* and ring *b* is taken up, and the degree of fineness of crushing is regulated.

For aiding proper lubrication of the anti-friction rollers *i* and gun-metal bush *o*, the lower portion of the pillar is made hollow and fitted with a couple of pipes *s*, leading from oil reservoirs (not shown) mounted on the frame, the pipes having branches *t* extending through lateral openings in the wall of the pillar *e* to just above the roller and bush respectively. The bearing *f* requires but little lubrication, as it is well protected from dust; but it is formed with a passage *u* leading from an external grease-cup *v*, closed by a screw plug *w*.

By the use of manganese steel a great economy in weight is effected, while the price of the machine is but triflingly increased.

*Dies.*—Various materials have been tried for securing the greatest durability in the dies. Many makers confine themselves to chilled white iron (cast), which has proved superior to ordinary steel. But it has a tendency to chip away, and it would seem that an intensely hard surface is not so desirable as toughness. Hence dies made of chrome steel and manganese steel have come greatly into favour of late. This class of casting, being of uniform thickness (thus avoiding stresses set up by uneven cooling), and requiring no tooling afterwards (the rough casting being all that is required), seems an ideal application for manganese steel, and the breaker dies turned out by Hadfields, Sheffield, have gained a high reputation. They have an additional advantage in great saving of weight, being cast quite thin as compared with iron.

The average wear of chilled cast-iron dies is stated by Prof. Louis at $\frac{1}{10}$ lb. per ton of stone broken.

The actual wear on cast-iron dies was particularly noted at the Lucknow mills, with the following results, computed on 10,321 t. broken. The swing-jaw dies used numbered 8; their weight new was 456 lb. each, old 392 lb., thus the loss was 64 lb. each $\times$ 8 = 512 lb. iron, or ·05 lb. per ton. The fixed-jaw dies used were 7; their weight new was 364 lb. each, old 308 lb., thus the loss was 56 lb. each $\times$ 7 = 392 lb. iron, or ·038 lb. per ton. The total wear was therefore ·088 lb. of iron per ton crushed, counting in that ore which passed the grizzly, or say 10%, which would bring the figure to about ·096 lb. per ton.

The actual loss by attrition and chipping in the case of manganese steel dies is not a fraction of this figure. But economy in this direction is of the falsest kind. To leave dies in wear for a longer time in order to reduce the cost per ton milled for dies worn, is to add very much more to the cost of subsequent operations by lessening the duty and increasing the wear and tear of the fine-crushing plant. It pays to replace breaker dies the moment their worn surfaces allow anything larger than 1½ in. to pass them. This was satisfactorily established by the author at Lucknow, where

accurate records of the stamp duties were kept, and the fluctuations in these figures coincided in a most remarkable manner with the wearing down and renewing of the breaker dies, showing variations up to nearly 20% when circumstances compelled undue delay in renewals.

The tendency to retain dies in wear after they should have been discarded is much more likely to occur with cast-iron than with manganese steel, both because of the labour entailed by their inconvenient weight, and because of the very large proportion of metal discarded as compared with that which is actually worn away. The tough steel allows of the dies being made hollow. In the case of gyratory crushers, Hadfields make both the cone mantles and the concaves in narrow sections, so that renewals can be effected of the parts actually worn without discarding the remainder, and thus an enormous economy in spare parts is brought about.

A point in connection with the corrugated dies made for swing-jaw breakers, that does not receive sufficient attention from some makers, is that the ribs and hollows in the opposing faces are not arranged to come into proper relationship. Obviously the ridge on one plate should be exactly opposite the furrow on the other; very often this is not the case, and it becomes impossible to set the machine at its proper gauge because the ribs meet each other. The spaces left in the corners in that event are absurdly large, and vitiate the whole operation.

An overhead crawl will suggest itself naturally as a most useful adjunct in removing and inserting dies.

*Sectional Machines.*—The abundant strength which is an absolute essential to the reliability of this class of machine—and an unreliable breaker is worse than none—makes it impossible that a satisfactory article can be turned out in sections for bolting together. The attempt has often been made, to overcome difficulties of transport, the heaviest piece not exceeding 3 cwt.; but to cope with the enormous strains created in breaking hard stone—and it is for hard stone that a breaker is needed—nothing less than a massive casting will suffice.

*Cost of Breaking.*—In but few mills are separate accounts kept of the cost of breaking and subsequent milling, because the work is so often done by the same engine. But Truscott quotes "just under 1*s.* per ton actually crushed" for Blake breakers at the Ferreira mill, and "about 13*d.* per ton actually crushed" for Gates breakers at the City and Suburban, both operating on Rand banket. The product of the latter is given at $1\frac{3}{4}$ in. The Bonanza Co. in their report for 1899 give the cost at 8·45*d.* per ton milled; and the official statement of the Ferreira Co. for 1898 is 4·4*d.* per ton.

Franklin White quotes 9·474*d.* per ton at Luipaard's Vlei.

At Waihi, N.Z., where the ore is "kiln-dried" (really burned), and water-power is employed, the cost of breaking by gyratory machine is stated (1896) at 8·14*d.* per ton, and that figure apparently does not include maintenance (renewal of wearing parts); in 1898, the cost was reported at 10·86*d.* per ton.

The following figures are derived from the author's reports to the Wentworth and Aladdin Cos., New South Wales:—

| Year. | Tons. | Maintenance. | Wages. | Fuel. | Total. |
|---|---|---|---|---|---|
| | | *d.* | *d.* | *d.* | *d.* |
| 1896 | 10321 | 2·172 | 2·669 | .. | .. |
| 1897 | 11562 | 2·355 | 2·382 | 4·000 | 8·737 |
| 1898 | 14823 | ·392 | 1·858 | 2·698 | 4·948 |
| 1899 | 14481 | 1·267 | 1·900 | 1·919 | 5·086 |
| Average | (51187) | 1·442 | 2·152 | 2·790 | 6·384 |

With regard to these statistics it may be said that they refer to jaw breakers, are calculated upon tons delivered to the batteries, and are based upon the number of trucks, one being weighed occasionally; there is therefore room for an error of possibly 10 % through trucks not being properly filled. A further 10 % may be added to the cost to allow for material which passes through the grizzlies, and consequently avoids

the breaker. The item maintenance is taken from the mine account books, and may be reckoned at about $1\frac{3}{4}d.$ per ton actually broken. Labour is computed at 1 man (day shift only) at 45*s.* a week for 51 weeks (excluding holidays), the battery output being 45 to 50 t. per 24 hours from 30 to 35 stamps. The wages of the two additional men, one loading from the breaker to the elevator, and one from the elevator to the battery bins, are not included, because their services are necessary whether a breaker is used or not. Fuel cost is arrived at somewhat arbitrarily by assuming it to be one-third of the total estimated to be consumed in raising steam for both breaker and battery engines ; this very fairly represents the relative figure. But the actual fuel bill for milling is difficult to arrive at, because there are five boilers supplying a number of engines, pumps, air-compressors, lathes, circular saws, dynamos, steam-hammers, fans, &c., all doing more or less irregular work, and it is probable that the estimate is somewhat below the mark. The chief fuel is good wood at 9*s.* a cord, with some coal at about 14*s.* 4*d.* per ton. Grizzlies and breakers are set at $1\frac{1}{2}$ in. The machine in use is a Jaques swing-jaw ; beside it, in reserve, is a Hope.

As the whole mill depends on the breaker, a reserve machine should always be provided, when operations are on a considerable scale.

**Fine Breaking.**—The principle of graduated reduction has been extended to the breakers themselves, and, when the establishment is sufficiently large, the work may well be divided between two machines, one reducing the stone to pass a 3-in. aperture, and the other taking it from that, a grizzly being sometimes interposed so as to avoid unnecessary crowding of the second machine. This is a highly commendable practice, and permits of the product from the second breaker being reduced beyond the common limit of $1\frac{1}{2}$ in. gauge. Indeed it will be found to pay well to break down to $\frac{1}{2}$ in., and feed the fine crushing machinery—stamps, rolls, &c.—with nothing larger than that. "Tandem crushing," i.e. one breaker feeding another, is growing in favour on such

progressive fields as the Rand, and "multiple" machines are made to effect the same purpose.

A good example of this type of machine is the Blake multiple-jaw, which, in combination with an ordinary breaker, can be used to obtain a product that will pass a 30-mesh wire screen, though the economical limit is said by the maker to be found between 14 and 20 mesh. In construction, the main sliding or toggle jaw is replaced by main swinging,

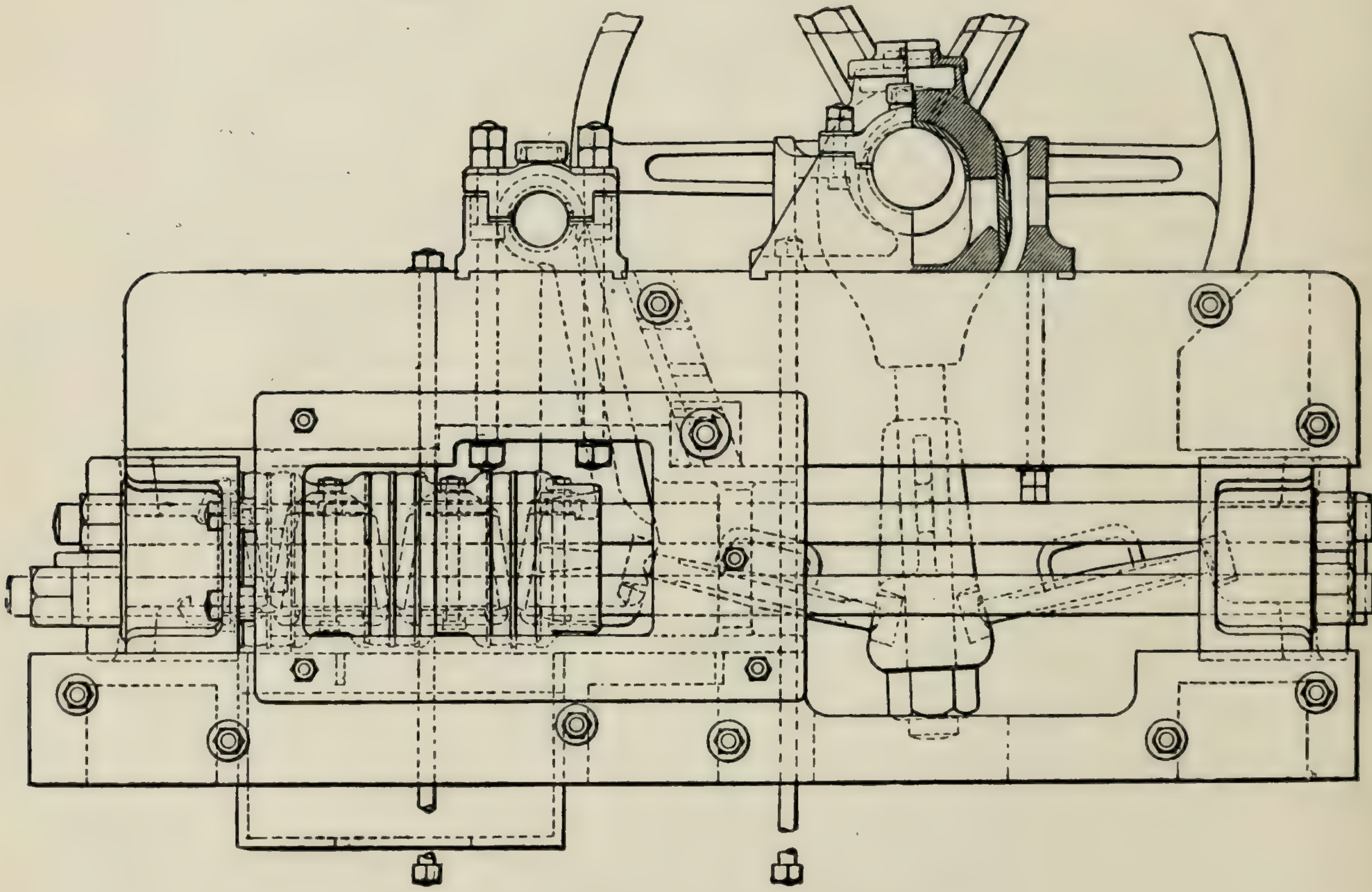


FIG. 19.—BLAKE MULTIPLE BREAKER.

thus doing away with the upward thrust on the tension rods and consequent wear. Experience has also shown that in machines of not over $\frac{1}{2}$ in. width of opening, it is better to use several small machines with a series of jaw openings (say 15 by $\frac{1}{2}$ in.), than a single large machine with a series of openings 24 or 36 × $\frac{1}{2}$ in. In an installation for the Haile mill (S. Carolina), dealing with tough coarse quartzite, the product of a 20 × 10-in. fed a 30 × 5-in., and the issue from this was split between two parallel sets of 60 × 2-in. multiples, each with 3 jaws; it left them at about $\frac{1}{4}$ in. gauge, and

passed thence to rolls for reduction to 40 mesh. At each stage, screens removed the fines as fast as made. A necessary

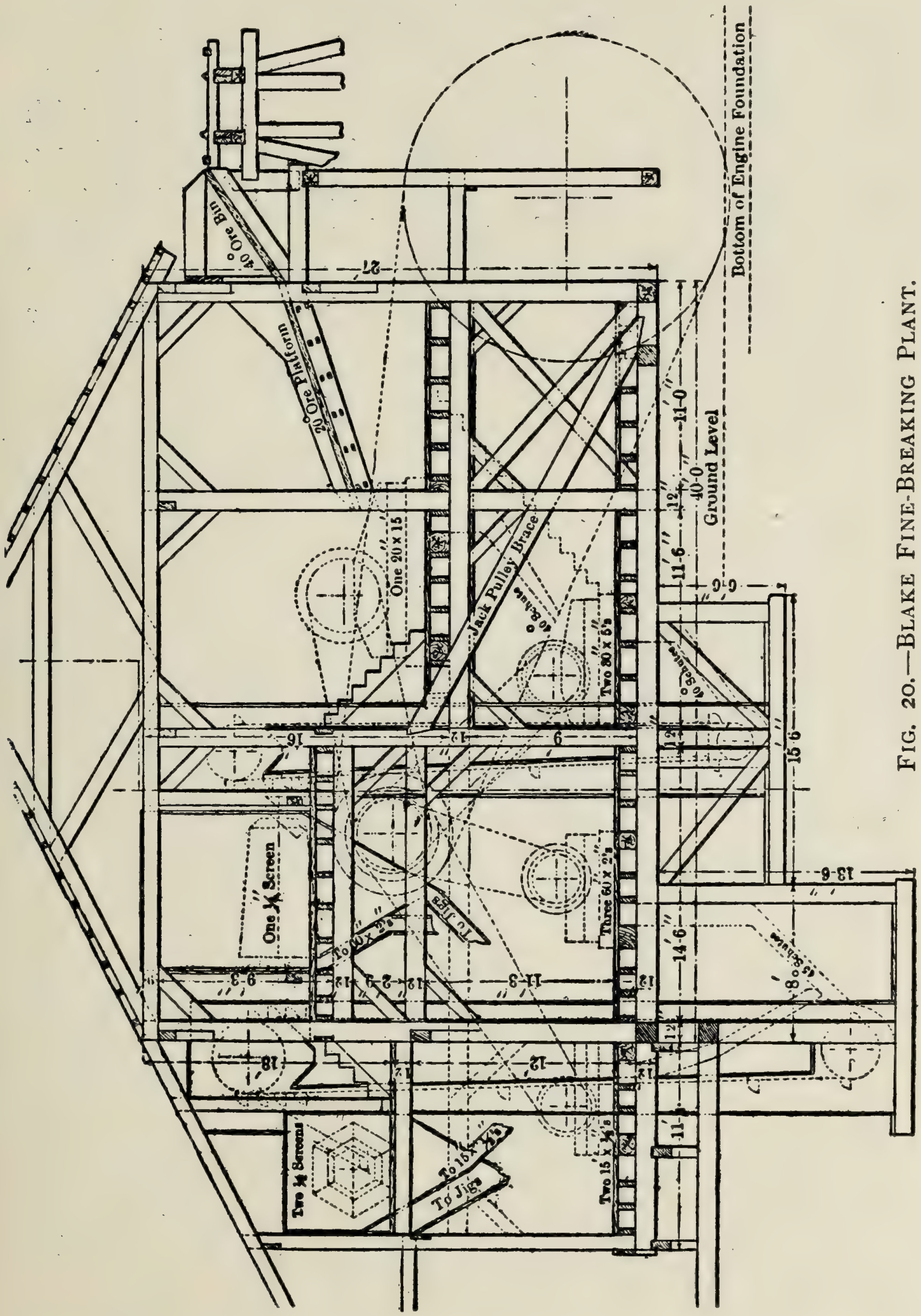



FIG. 20.—BLAKE FINE-BREAKING PLANT.

condition is dry ore. The 60 × 2-in. multiple is shown in Fig. 19.

A complete plant for reducing 600 to 700 t. daily from 15 in. down to $\frac{1}{4}$ in. is illustrated in Fig. 20. The ore arrives by side-tipping trucks carrying $7\frac{1}{2}$ to 8 t. each, and first enters two 20 × 15-in., which reduce it to $2\frac{1}{2}$ in. gauge; each of these delivers to two 30 × 5-in., giving a $1\frac{1}{2}$-in. product; and this is divided between three 60 × 2-in. multiples, each with three jaw openings 20 × 2 in., the finished article passing a $\frac{5}{16}$-in screen. All rejections taken out by screens at each stage are elevated back for re-crushing. The final product goes to jigs. The cost per ton on 137,551 t., including jigging, notwithstanding that the ore was often damp or frozen, was:—

| | |
|---|---|
| Fuel for power | 3·125*d.* |
| Labour | 7·589*d.* |
| Oil, waste, etc. | ·893*d.* |
| Renewals and repairs | 3·571*d.* |
| Total | 1*s.* 3·178*d.* |

The Gates Iron Works have recently introduced a fine crusher, called "Style H," which is calculated to take the material which has passed an ordinary gyratory breaker, or a jaw crusher, say 3 in. ring size, and is capable of reducing same to such size that 90% or more will pass a $\frac{1}{2}$-in. screen. Economical wear of the dies is accomplished by using a movable ring for holding the concaves (which are only 7 in. long), and setting them perpendicularly, so that when the lower end becomes badly worn (the head having been raised to the full limit and the material becoming too coarse), the head may be lowered and the rings and concaves reversed, thus using what was the upper end, and which has received slight wear. The head in this machine, which is very large and only 10 in. in length, allows of making a cast-iron core or centre, which is fastened permanently on the shaft, and has an abrupt taper, so that a thin parallel shell may be drawn on with bolts, in the same manner that shells are used in rolls. The concaves may be set in in the concave ring by the use of a narrow concave, and both ends used again. It is therefore possible to get such close wear from both head and

concaves that only a comparatively small amount of metal is thrown on to the scrap-heap. A top shell is dispensed with, and the spider is bolted directly to the lower shell or frame. It is only a short job to remove the holding-down bolts, the clamping-bolts, the spider, and the ring, to reverse the concaves or insert a new set. The ring, being light, is easily taken to a convenient place to "zinc in" new concaves; and the expense of an extra ring, which can be fitted with concaves to be used when occasion requires, will further shorten the time the machine must be idle for such repairs. The discharge diaphragm is made independent of the frame, and can therefore be shifted while in place so as to discharge in front or at either side, which is often a great convenience. Fastened to the shaft, and gyrating with it, is a dust and water shield having a projecting flange, which enters a reservoir and is immersed in water or oil, thus forming a complete seal for the actuating mechanism, protecting it whether the machine is being used for wet or dry crushing. The capacity of the machine, on very hard porphyry, screened of all finer than 1½ in., is 4 t. per hour to give 90 % passing a ⅝-in. screen, and 1½ to 2 t. per hour passing ½ in., requiring 10 to 15 i.h.p., according to the makers. The machine weighs 5¼ t., has a speed of 600 rev. a minute, and its driving pulley is 24 in. diam. and 4 in. face. The ore must not, of course, be wet or sticky.

# CHAPTER III.

## WET MILLING—STAMPS.

NOTWITHSTANDING the continual and unceasing invention of new forms of pulverising or fine crushing machinery, the stamp battery retains its pre-eminence. It is often described as a crude and cumbrous machine, and in one sense perhaps this is true—there is an absence of what may be called fine engineering about it, and it is essentially a simple, a rough-and-ready implement. Probably it is to this circumstance that its survival is largely due. Devoid of intricate mechanism and of those delicate contrivances which involve constant care and attention, its component parts are strong and tolerant of ill-usage, besides being uncostly and easy of renewal. Moreover, the ordinary conditions of wear and tear do not involve complete suspension of work, as in most machines ; and it is easy to increase or diminish the crushing capacity, by adding to the installation in gradual stages, or by suspending so much of it as is temporarily not needed, and thereby avoiding waste of power and useless wear. It will be a remarkably clever invention that is to supersede the gravitation stamp battery as turned out by the foremost engineering firms to-day.

**Site.**—A primary consideration in the erection of stamps is a suitable site. This seems self-evident enough, yet observation shows that it is very commonly neglected, to the hindrance of economic milling. The conditions to be studied are :—

1. Source of motive power and its application.
2. Conveyance of ore into the battery.
3. Provision of necessary water supply.
4. Automatic passage of ore through the various stages of treatment.
5. Ultimate disposal of the tailings.

Water power being, when constant and reliable, the cheapest motor, there has been a disposition in some cases to select a site almost solely with reference to this question, quite regardless of the facilities which exist for transmission of power (by electric cable) over great distances. Power is a heavy item in milling costs. When steam is used, the location of the boilers has a controlling influence. Steam cannot be carried far without great loss, and it should never be transmitted down hill; being costly in itself, it will not bear the additional burden of electric transmission. Supply and delivery of fuel (coal or wood) are prominent factors, and of water not much less so, while disposal of ashes is often entirely forgotten until their accumulation becomes a nuisance, and their removal an expense.

Conveyance of ore into the battery is subject to various conditions, as for instance whether it is to come from the mine direct, and, if so, whether from more than one shaft; whether public or custom milling is contemplated; whether sorting on surface is to be practised; and whether hand breaking or machine breaking is adopted, and, if the latter, whether the breaker is to be at the shaft-head, in the battery, or on some other site. Transport may be aerial or terrestrial, and automatic, or by human, animal, or mechanical traction, according to governing conditions; but whatever form it may take, it should be as direct as possible, and avoid handling.

In considering water supply, it must be remembered that wet milling demands 7 to 10 t. of water—as clean as can be got—for every ton of ore put through. It would thus appear at first sight as if immediate contiguity to water were a prime necessity. But, owing to its fluidity, the transport of water on a decline costs almost nil, and its elevation by suitable machinery, such as steam pumps, is a very small figure compared with a similar proceeding in the case of ore.

That the passage of the ore through the various stages of treatment shall be automatic, or in other words, that the gravitation of the water used in its treatment shall form the vehicle for its conveyance, is a matter that cannot be overlooked. Before planning a battery, therefore, it is above all

things wise to have determined upon every step in the treatment. Frequently it is unavoidable that there shall be one elevation of the ore into the battery; but there is no excuse for rendering a second lifting necessary, more especially as there will be the added weight of water to be raised also.

Lastly, the accommodation of the ultimate residues or tailings must be provided for; and when it is borne in mind that they are practically about double the ore mined, both in volume (by the multiplied particles) and in weight (by added water), it will be seen that they must not be ignored. Yet they frequently are, and it is not an unknown occurrence that batteries have been virtually blocked up with tailings. In America, the mining industry often enjoys a sort of privilege, and batteries are allowed to pour their tailings into flowing streams; but that is not the case in all countries, and it should be so in none. The sludge question is a vital one on many fields.

To sum up the various considerations, it may be said that the minimising of transportation of the ore by concentrating the crushing machinery at the principal shafts is a main consideration; and that a fairly steep hillside, with a fall of, say, 1 ft. in 1½ to 2, offers exceptional advantages in construction. There are fields, however—of which the Rand is a notable example—where this condition can hardly ever be secured, no sites having an inclination of more than a few feet in a hundred. This entails as a rule increased elevation of the ore, and always increased cost of structure. The only indispensable feature for Rand batteries is an abundant space for disposal of tailing sands and for impounding and settling tailing slimes. In choosing a hillside above a stream, flood levels must be ascertained and carefully avoided.

**Foundation.**—The very essence of a stamp battery is a good foundation. Without it, no battery can run at its full speed and capacity for any length of time, and indeed it will soon shake itself to pieces, necessitating constant delays for repairs. With the heavy stamps and high speeds favoured by modern practice, the importance of a solid foundation is

intensified, and no consideration should be allowed to stand in its way.

At one time, the only substructure regarded as proper for supporting the mortar was made up of timber, but latterly concrete is coming into vogue. In any case, excavation in firm ground must be provided for reception of the mortar block, whether timber or concrete. Sometimes the conditions are such that this trenching is entirely in solid rock, and when the character of the rock is not such as to unduly exaggerate the cost, this is to be preferred, always remembering that the space must be sufficient to allow about 1 to 2 ft. all round the blocks for introduction of packing. The lowest part of this trench at least must be in solid rock, or concrete must be run in as a substitute.

Timber is still the more common material for battery foundations. Nothing but sound heart wood is admissible. In America, the favourite kinds are Oregon pine (which, though light, is durable), and pitch pine. Australian builders make use of several sorts of eucalyptus, especially karri and ironbark ; both are extremely durable, but very heavy and hard to work. Africa, being dependent on imported timber, employs both American and Australian. These logs or piles are usually 12 to 16 ft. long, according to the nature of the ground and the proposed height of discharge from the mortar. They are ranged side by side and bolted together longitudinally in groups forming a solid mass having a cross section of 4 to 5 ft. by 20 to 30 in. When trees can be got of a size to give sound timber 2 ft. sq., the building up is simplified, as 3 logs will suffice for an ordinary mortar ; but when only stuff of lesser dimensions is available, much more labour is entailed in building up, and more bolts and cleats are required. It has even been recommended that mortar blocks should be built up of 2- or 3-in. planks, tarred and spiked together ; but it can only be under most exceptional conditions of paucity of sound timber that such a troublesome proceeding is to be entertained, and in such a case concrete would be a better solution of the difficulty. Australian ironbark wiil remain as good as new for 50 years, and probably much longer.

The construction of a timber mortar block as adopted by the author in New South Wales is shown in Figs. 21 and 22. The three ironbark logs *a* are each 12 ft. 6 in. long, 26 in. wide, and about 19 in. thick, so that when fastened together by 1½-in. tie-bolts *d*, they present a top surface measuring 4 ft. 8½ in. long and 26 in. wide, corresponding with the base of the mortar box. At their lower extremity they are further united by longitudinal tie-timbers *b*, 12 in. sq., recessed as shown, and secured by 1½-in. tie-rods *c*. Thus the base of the structure is widened to the full extent of accommodation of the trench, the walls of which are seen at *e*. In this instance, the lower part of the excavation was in solid rock, so that neither concrete foundation nor masonry retaining walls were necessary. When in place, the space around the vertical blocks was filled and tightly rammed with earth and coarse tailings. The longitudinal ground sills *f* measure 14 in. deep and 12 in. wide; being 14 ft. long, they serve for two 5-stamp batteries. They are held in place by 1¼-in. bolts *g*, leaded in solid rock. Upon them rest the transverse sills *h* of the battery frame; these are 17 ft. long, 15 in. deep, and 9 in. wide in the case of the outside ones, whilst the centre one is 12 in. wide. They are secured to the longitudinal sills *f* by 1¼-in. tie-bolts *i*. The battery frame posts *k* are built up from the sills *h*, and carry the cam-shaft *l* and the guide timber *m*. The mortar box *n* is laid on the mortar blocks *a*, and is kept in place by the bolts *o*.

Though made of ironbark, which is marvellously durable, all the underground timbers were well dressed with boiling tar after being trimmed to accurate dimensions.

The longitudinal tie-timbers *b* are sometimes replaced by a plate of ⅝-in. iron, and when the mortar blocks have to be built of several pieces of timber of smaller girth, this is perhaps a superior method; in that case two or more tie-plates may be used, and they are even added to the tie-timbers.

It is most essential that the mortar blocks shall be very firmly packed in place, and that when finished they shall be truly vertical and provide an absolutely true and level face for reception of the mortar. Rather than run any risks of

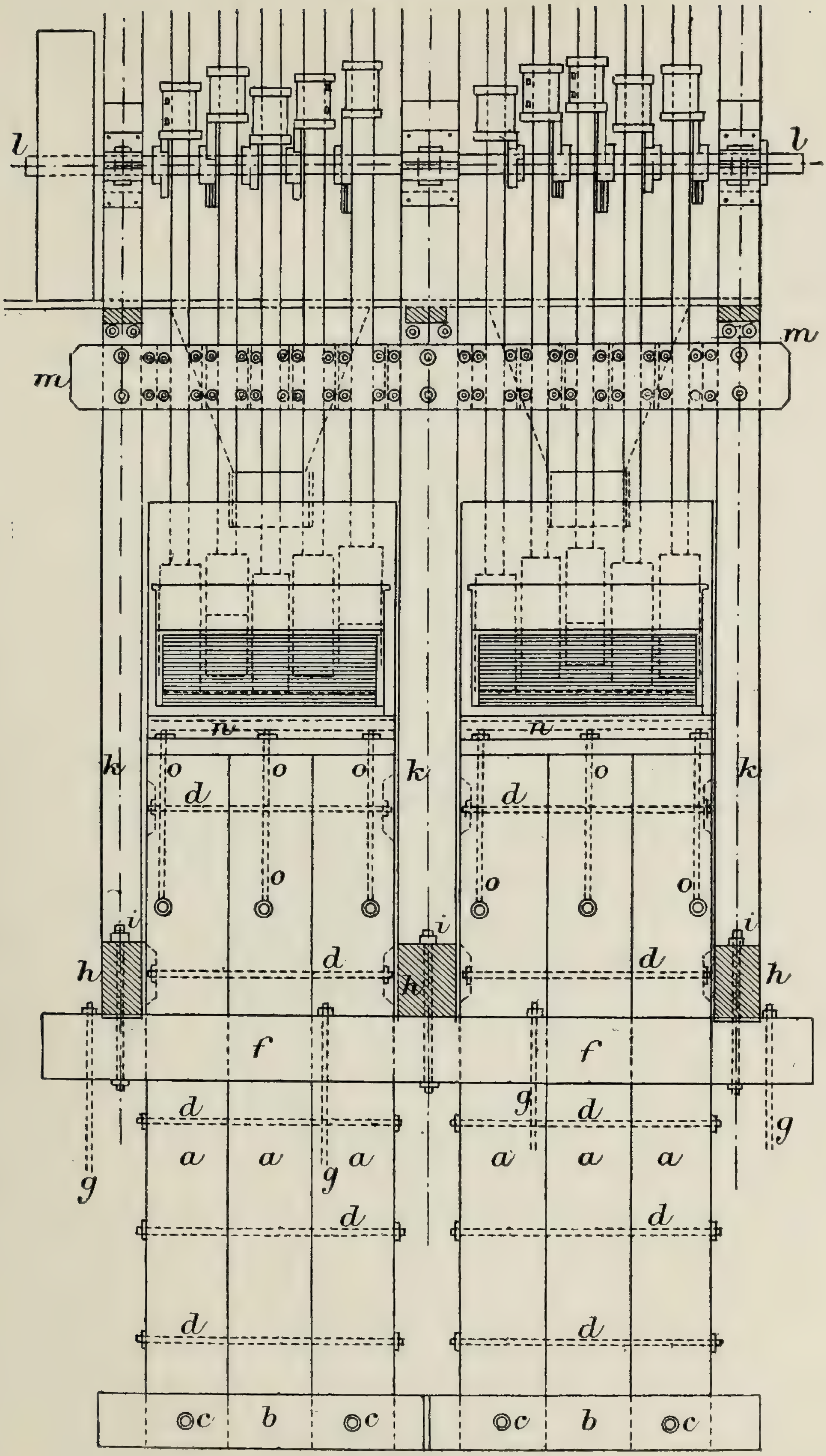



FIG. 21.—BATTERY FOUNDATION.

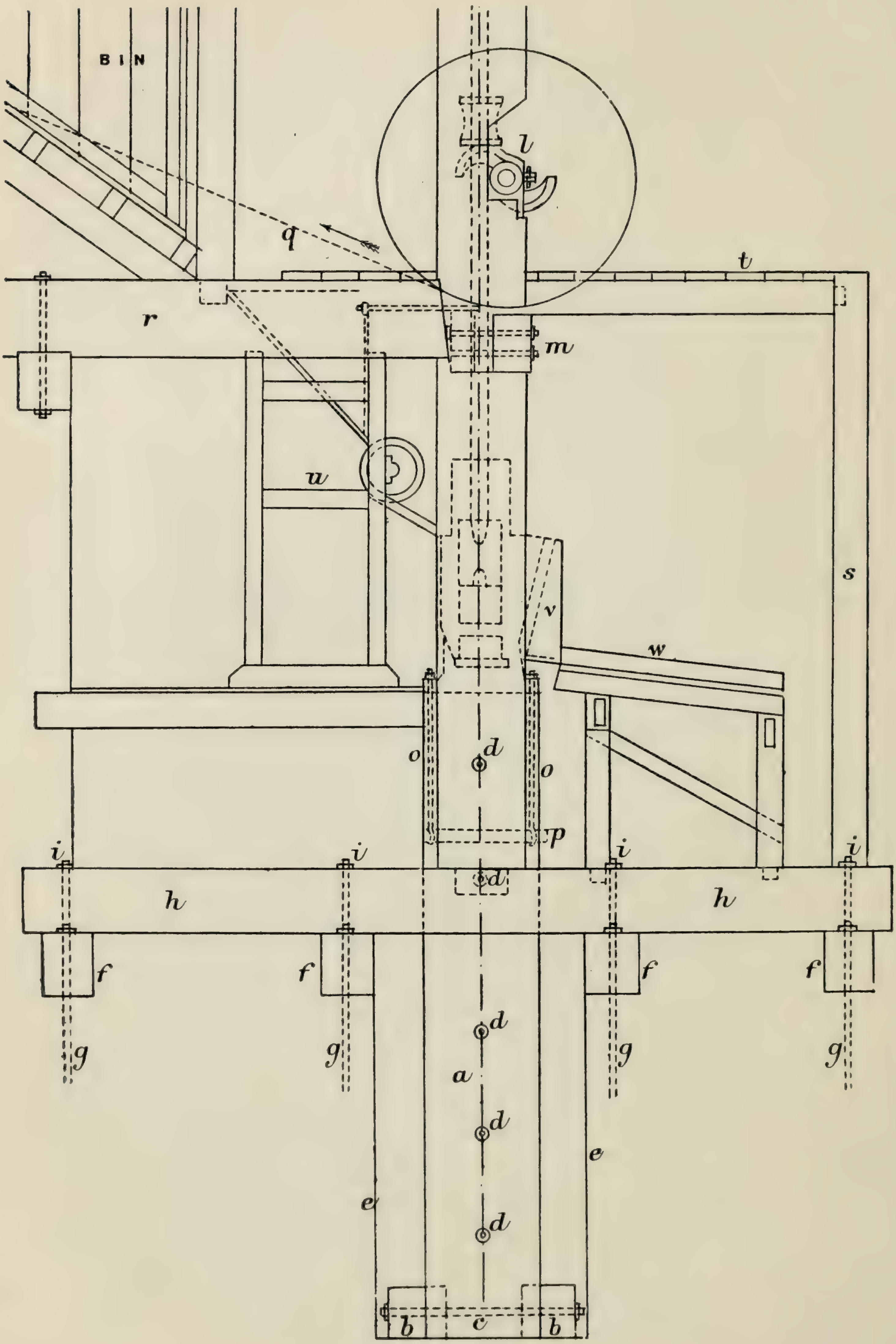



FIG. 22.—BATTERY FOUNDATION AND FRAMING.

the face becoming rounded, it should be very slightly dished.

The method of securing the holding-down bolts *o* here shown, is an improvement on the ordinary practice. Usually the bolt is passed down a hole bored to receive it, a hand-hole being cut at the bottom to enable a washer and nut to be introduced and fixed. After a few years, it will be usually quite impossible to shift this tie-bolt, and, should it become necessary to replace the mortar, great difficulty will be experienced in moving it, and the bolts will be rendered useless. By the plan shown, which was introduced by Mr. Robert Canning, chief engineer at Lucknow, the rods *o* occupy shallow grooves in the blocks *a*, and are held at the lower end by loops, through which is passed a short length of 1½-in. gas-pipe *p*. They serve the purpose admirably, and can be removed at will without risk of any damage.

The face of the mortar block must be well protected while it is waiting for the mortar. A cushion of some sort is always introduced between the wooden face and the iron mortar. Sometimes this takes the form of two or three thicknesses of blanket, thoroughly tarred on both sides; and sometimes sheet rubber ¼ to ⅜ in. thick is preferred: old vanner belts answer admirably for the purpose. The object is to minimise the "jar" caused by the stamps when in operation, and thereby avoid any movement of the mortar, which might admit dust or pulp, and bring about a disturbance of level.

Most of the Rand batteries * have erected wooden mortar blocks on concrete foundations, even though the bottom was sandstone. At the City and Suburban Mill, a 12-ft. deep trench was given a bed of concrete 8 ft. long and 3 ft. wide for each battery; the surface was made quite true, and on it the mortar blocks were placed, 15 pieces being bolted together for each box; and 2-ft. masonry walls enclose the whole, spaces being tamped hard with tailings. The Wolhuter has an 18-in. concrete bed, and the Simmer and Jack a 24-in.; at the latter, each block consists of 8 timbers, 14½ in. sq.

When suitable timber cannot be had at reasonable cost,

* Hatch and Chalmers, p. 186.

concrete is a most acceptable substitute, and is much more durable so far as resistance to water and vermin is concerned. Provision must be made in building for the various holding-down bolts, and these latter should be furnished with broad stout iron washers, so as to distribute the strain as much as possible. Such a block will be topped with a capping of at least 6 in. of hard timber, such as karri, ironbark, teak, or oak.

The Geldenhuis Deep mortar blocks consist of massive concrete, each set of 10 head having a separate basis. This innovation was forced upon the builder by the soft and yielding nature of the ground, compelling a large area to be covered with a solid and substantial foundation. It would seem that no timber is used under the box.

At several of the immense Alaskan mills a very heavy concrete foundation, about 8 ft. wide at the base, extends the entire length of the battery. Upon this, under each mortar box, is placed a cast-iron anvil block weighing about 7 t., bolted securely to the concrete foundation, and planed on top to receive the mortar. In that climate, pine timber blocks do not last more than 10 years.

**Framing.**—It will be manifest from Fig. 21 (p. 51) that the main duty of the battery frame is to support the cam shaft *l* and guide timber *m*, so that they will successfully withstand the incessant vibration caused by the falling stamps, and the tendency of the driving belts or gear to pull or thrust the structure out of the perpendicular.

*Timber.*—It is, therefore, above all things necessary to secure sound, straight, and well seasoned timber for the purpose. Heaviness is not essential so long as the requisite strength and rigidity are there. Most countries produce a suitable growth of wood, and nearly all varieties of pine are more or less well adapted. Thus in Australia, the native cypress, though somewhat heavy, is not difficult to work, and, where not exposed to the sun, does not warp or shrink, and thus answers very well. New Zealand has two excellent woods in rimu and kauri, the former being preferable, as the latter is heavy, and liable to longitudinal contraction if not

very well seasoned. Californian builders use indiscriminately the local red spruce, sugar pine, and yellow pine, all possessing the necessary qualifications. In the more northern States, such as Dakota, Montana, and Idaho, and for the immense mills of Alaska, the well known Oregon pine is everywhere favoured. This magnificent timber is also employed almost universally in South Africa, and to a large extent in Australia, its lightness combined with strength making it preferable to most, when an imported wood is necessary. Pitch pine is sometimes employed, but it is heavy, and very inflammable. European pines (Norway and Dantzig) are used by English builders, and a certain amount of Alaska spruce locally; while in Canada preference is given to best "Southern pine" (long-leaved yellow pine) which "never checks nor seasons to cause disturbance." *

Posts.—The most common arrangement of mills is to have two 5-stamp batteries in the same frame and sharing the same cam shaft, though occasionally each 5-head is separately driven; and it has become a stereotyped practice to have the central frame post *k* (Fig. 21) of much greater dimensions than the two outside posts. This practice would seem to have arisen from the fact that when two 5-stamp frames are united, the centre post will consist of an outside post of each of the batteries. The necessity for a strengthening of the central post is certainly not apparent; in fact it would seem theoretically to be the least important of all. The greatest strain must undoubtedly be on the post next the driving pulley, and that should be of additional strength if any. The other end post even is more important than the central one, for the latter has simply to act as a preventive of sag in the middle of the cam shaft. The solidity and rigidity of the structure depend on the outside posts and not on the central one.

Struts and Stays.—The provision of struts and stay-bolts will depend upon the direction in which the pulling strain of the driving belt or the thrust of gearing is exercised. In the example shown in Fig. 21, it will be seen that both struts and ties are dispensed with. This is because the pull of the

* J. E. Hardman, Jl. Fed. Can. Min. Inst., ii. 104 (1897).

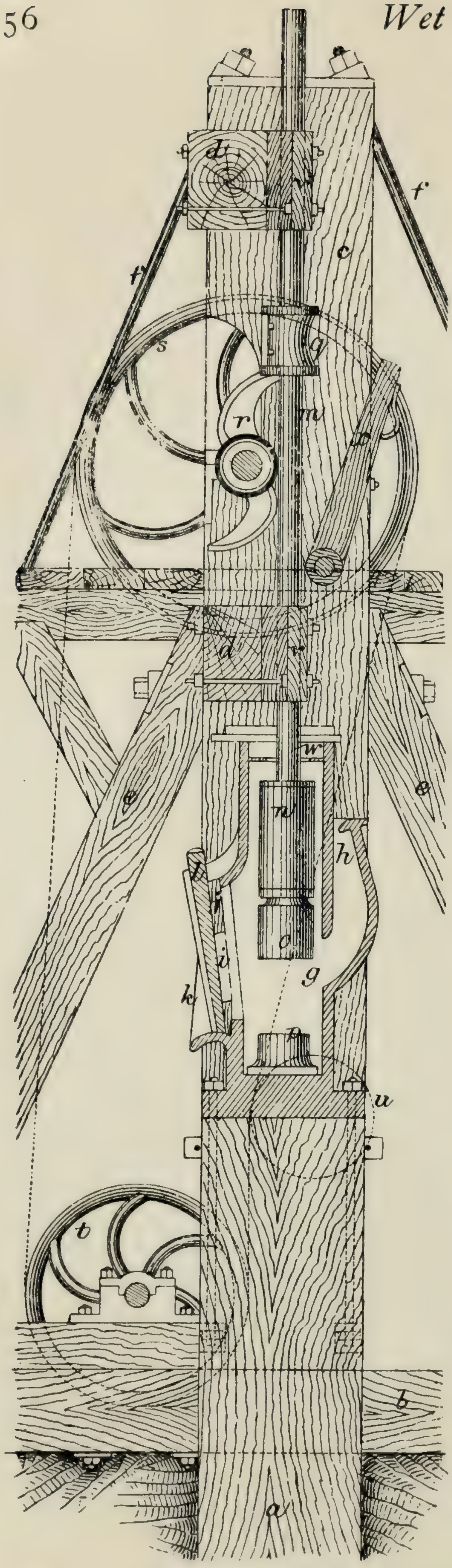



FIG. 23.—BATTERY WITH A FRAME.

driving belt *q* is in the direction of the arrow, and is effectually resisted by the lower beams *r* of the bin, which measure 18 by 9 in., and by central beams 15 by 9 in., which carry the rock-breaker (see Fig. 11, p. 20). The frame posts are sufficiently tied together longitudinally by the lower guide timber *m*, and by an upper guide timber near the top (not shown in Fig. 21). In this case a light timber (8 in. sq.) framework *s* carries the 2-in. cam floor *t*, and a similar structure supports the self-feeder *u*. The advantages of this arrangement are numerous. Not only is considerable economy effected in the saving of strut timbers and stay-bolts, but the driving belt *q* is well out of the way of all men engaged in operating or repairing; the self-feeder *u* and the cam shaft *l* are accessible from all sides without risk of entanglement in belts; no belt tightener is needed with the ample length of belt provided, and if it were it could be conveniently added at the end of the bin; and the disadvantage

of the discharge *v* and plate table *w* being overshadowed and darkened by the cam floor *t* is reduced to a minimum. Though it is not always possible to contrive that the strain of the driving belt shall be upwards, as in this instance, it should always be towards the feed side of the battery, as on that side is the thrust of the bin to counteract it, and this principle will be found carried out in all the large modern mills both in America and in South Africa.

When bins and self-feeders are not used—which reprehensible practice still survives in many places—struts and stays must be provided. An example of the *A* frame is given in Fig. 23 (scale $\frac{3}{8}$ in. = 1 ft.): *a*, foundation timber or mortar block; *b*, transverse sills; *c*, frame posts or battery posts; *d*, guide timbers; *e*, struts; *f*, stay-bolts; *g*, mortar; *h*, feed opening; *i*, discharge screen or grating; *j*, screen-frame; *k*, lugs for screen-frame; *l*, wedge holding screen-frame; *m*, stamp stem; *n*, stamp head; *o*, shoe; *p*, die; *q*, tappet; *r*, cam; *s*, cam-shaft pulley; *t*, driving pulley; *u*, tightener; *v*, guides; *w*, battery covers; *x*, prop or finger post for stamp when idle. The extra complication of this arrangement is visible at a glance. Besides the additional cost, access to the battery both for operating and repairing is impeded on all sides, and the absurdly short centres for the driving belt necessitate a tightener, which will make the pulley *t* exert an undue strain upon the cap of the bearing. Such a system is excusable only in a hand-fed battery of light stamps.

Another form of framing, giving considerable scope in the selection of a site for the driving shaft (counter-shaft or lay-shaft), which has come into very general use, is the knee frame shown in Fig. 24. The mortar blocks or piles *a* are 30 in. sq., 12 to 14 ft. long, and bolted together by six $1\frac{1}{2}$-in. bolts and nuts. The longitudinal ties or foot timbers are 18 in. sq. and 6 ft. long, are recessed 6 in. into the piles *a*, and are held together by six $1\frac{1}{8}$-in. bolts. The longitudinal battery sills *b* measure 28 ft. long and 18 by 24 in. diam.; they lie parallel with the cam shaft, one being 5 ft. between centres behind the mortar block, a second 5 ft. in front, and the third 14 ft. from the second. They are held down by bolts 8 ft. long, keyed

into masonry or bedrock. When the work has proceeded so far, the outside transverse sills *c*, 28 ft. long and 20 by 14 in., are wedged into the battery sills, and secured by bolts. The centre line timbers are 28 ft. long and 20 in. sq. The intermediate line timbers are 28 ft. long, by 20 by 14 in., are dressed on the upper side, and are reduced to $19\frac{1}{2}$ in. and $13\frac{1}{2}$ in. where they pass the battery blocks ; they are let 3 in. into the sills, and are secured by keys driven both ways and

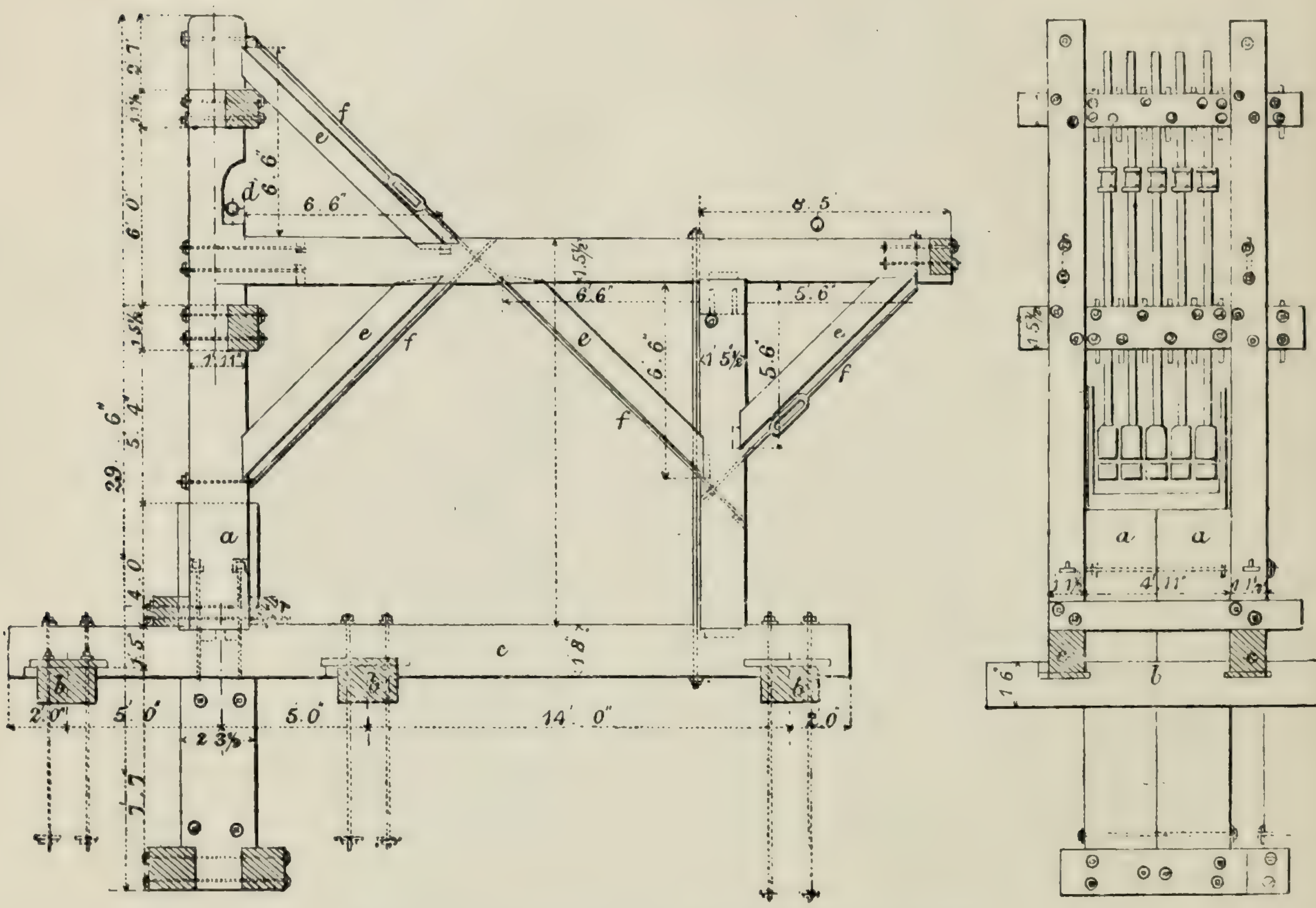


Fig. 24.—Details of Battery Framing.

by 2 iron bolts 33 by $1\frac{1}{8}$ in. The outside battery post is 23 by $13\frac{1}{2}$ in., and is let into the sills. The posts are secured to the transverse sills by 2 joint bolts 44 by 1 in., and are held together by the guide timbers *d*. A seat for the cam shaft journal is cut into the battery posts as at *d*. The bracing or struts *e* and stay-bolts *f* are on an extensive scale, necessitating a very substantial post just within the third sill *b*, and a stoutly framed cam floor. This arrangement may be justifiable when the builder does not know where the counter shaft is going to

be accommodated, because it prepares for withstanding strains from almost any possible position. Thus the countershaft may lie on the transverse sills *c*, just behind the battery post, and either before or behind the cam floor post; or it may be carried, as is most usual with this style of frame, on the cam floor itself, as indicated by the little circle.

But there is nothing to recommend this costly and cumbrous system. The man who in ordering a battery cannot specify how it is going to be driven is not worth catering for; and it is deserving of a great effort to arrange that the countershaft shall be on the feed side of the battery, as in Fig. 22. It will be seen from Fig. 25 (reproduced from a photograph of the Wentworth battery, Lucknow, N.S.W.) that the needless consumption of timber is enormous. The square timber, ranging from 18 in. for the posts and platform to 9 in. for the struts, will, in a 10-stamp battery, amount to 36 ft. run for posts, 75 ft. for cam floor, and 120 ft. for struts; to this has to be added the great area of 2-in. plank required to cover a floor reaching far out beyond the ordinary length of copper-plate table. The total weight of timber employed in such a frame will be very little short of 10 t., reckoning on Oregon pine, and much more if of cypress or of pitch pine. Then, as will be noticed, the passage of the millman from one cam floor to another is impeded by the driving belts, and he is always subjected to unnecessary risks in threading his way through moving belts and over little gangways. With such short centres, tighteners are a necessity, entailing further outlay, more obstruction, and additional bearings to watch. Further, the whole discharge from the battery and the copper-plate tables are in a dense shadow. Even if the complicated structure did its duty thoroughly in the way of securing rigidity and perpendicularity, it would not be commendable. But at Lucknow, in the author's experience, it did not. With a view, doubtless, of saving the services of a special engine-driver, the engine was placed inside the battery building, and the only site available for it was in front of the battery. Thus the pull of all the belting was in the one direction, and the effect of this was intensified by the thrust of the bins (from the constant

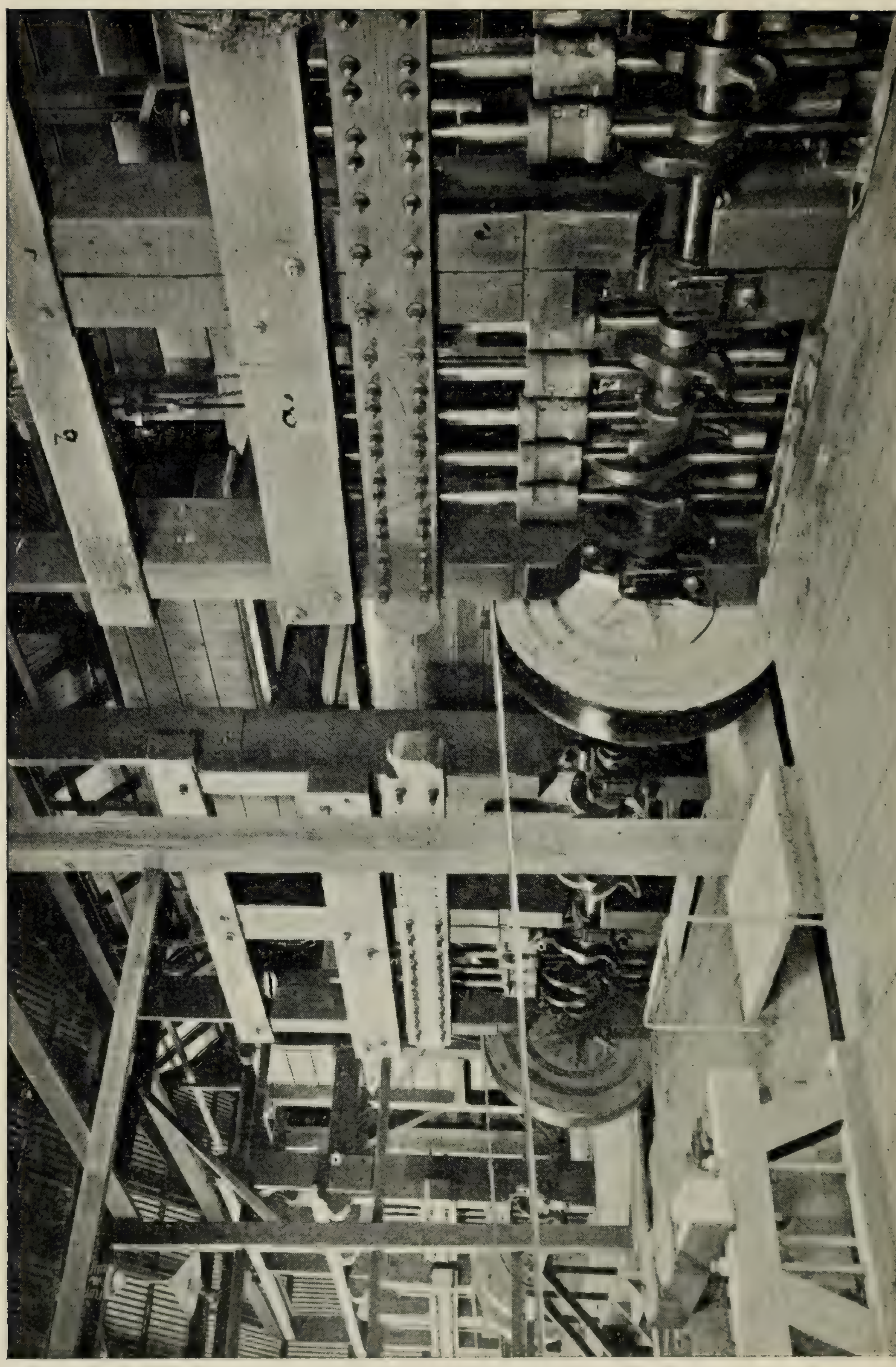


Fig. 25.—Battery Framing and Cam Floor.

and repeated fall of the ore towards the front) being in the same direction. When it became necessary, after about 3 years' operations, to increase the capacity of the battery from 20 stamps to 35, it was found that the whole of the old structure had been pulled several inches out of line. In the newly added batteries, the heavy tie timbers *a* (Fig. 25) were dispensed with entirely, and the top tie timbers *b* were much reduced ; the second central post *c*, which, as will be seen, does no work, was also omitted. No detriment whatever was incurred by these economies.

In marked contrast to this inconvenient arrangement is the style adopted in most of the Transvaal mills, as shown in Fig. 26. Here the structure is of the simplest and strongest, the cam floor does not obscure the screens and plates, the driving gear is out of the way of the operator, and a flood of light from windows and electric lamps is thrown upon the working parts which demand most constant attention.

It goes without saying that all wood joints must be mortised and tenoned in a proper manner, just as in all other construction, and stay bolts should have a coarse thread and heavy nut, with as broad a plate as possible under the head as well as under the nut. A good coat of tar (boiling hot and some pitch added), or two or three coats of red lead or iron oxide paint, should be given to all battery framing. In some cases lime wash is preferred, as contributing to the lightness of the interior of the battery house.

*Iron and Steel.*—While wood has proved itself superior to any other material for battery frames, inasmuch as it provides the needful rigidity and does not spring, there are cases where timber cannot be used, especially in countries that are cursed with the white ant. In this age of steel, there is a natural inclination on the part of the engineer who is building a battery to substitute his favourite metal for wood wherever he can do so—he profits directly from the additional iron or steel structure, while he would gain nothing from the timber work that would otherwise be used. Hence many iron and steel framed batteries are to be found in localities where excellent timber is available, and where no objection can be urged against it.

But the elasticity of these metals renders them distinctly inferior to timber. Very many of the batteries in Australia have iron or steel frames, but the weight of stamp and running speed there in vogue belong to a past age, and nobody looks to Antipodean batteries for an idea. On the other hand there

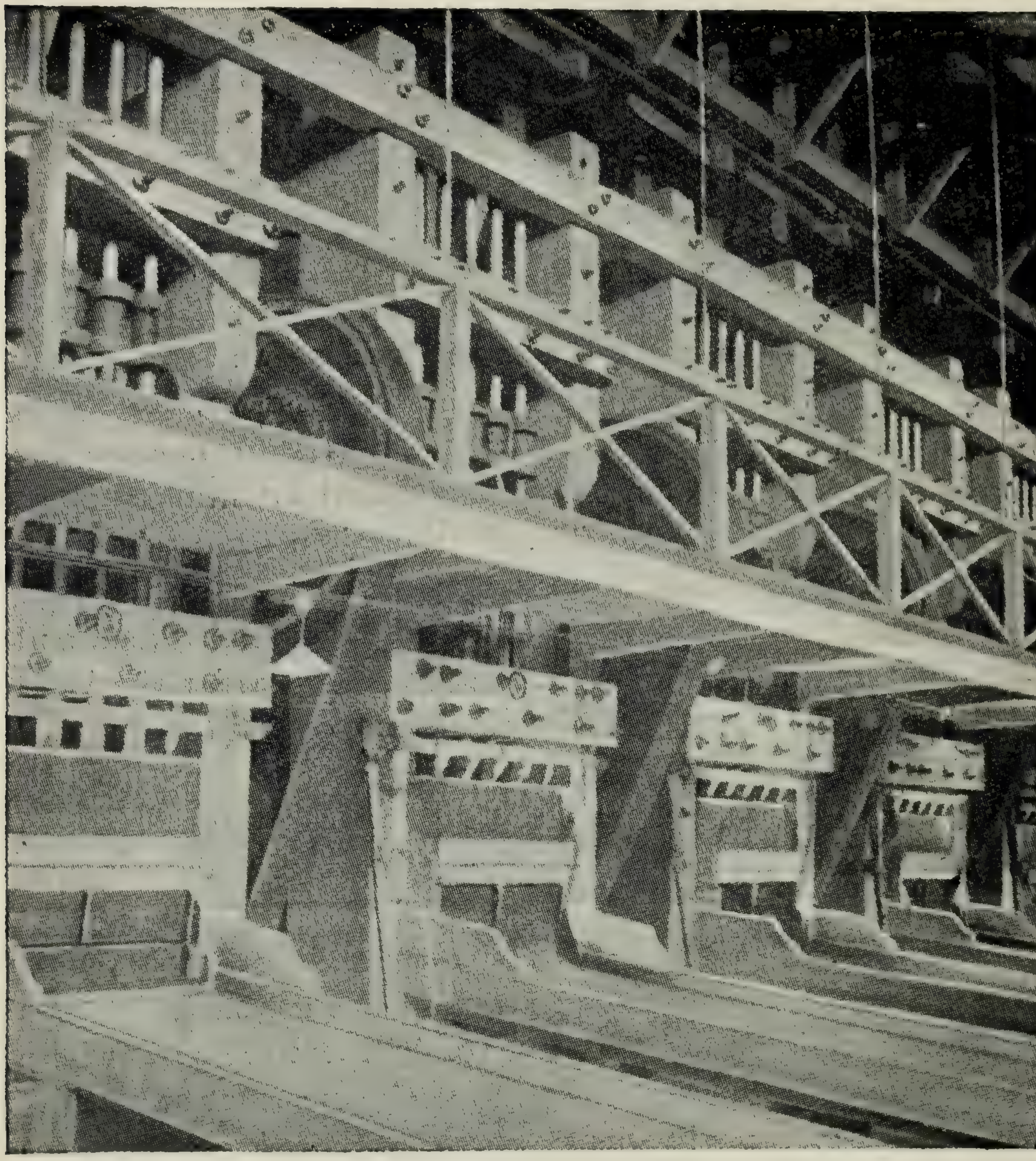


FIG. 26.—BATTERY FRAMING.

are instances on the Rand of iron-framed batteries being dismantled and re-framed in wood—involving delay and outlay which would not be incurred without very good reasons. The advantages of the metal structure are simplicity in erection and freedom from shrinkage or warping.

Cast Iron.—The design of iron and steel frames varies perhaps even more than that of timber frames. A favourite

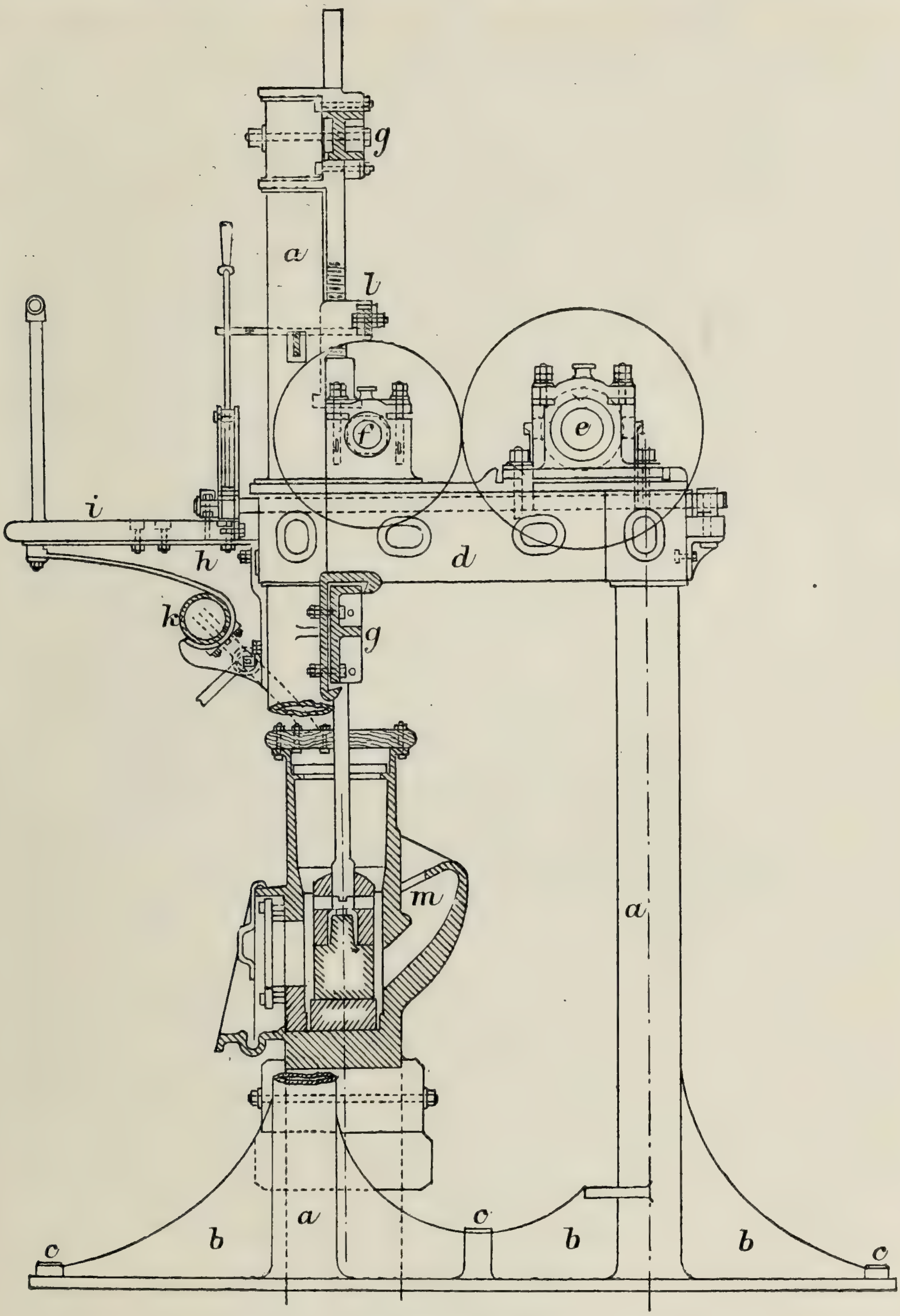



FIG. 27.—CAST-IRON BATTERY FRAME.

Australian pattern of cast iron frame is shown in Fig. 27. It consists essentially of hollow cast iron standards *a*, 10½ in. diam., the rear pair standing 49 in. (centres) from the stamp

stem. These columns are suitably flanged at foot *b*, so as to take powerful holding-down bolts *c*. They support a rectangular frame *d*, also of cast iron, which bears the heavy countershaft *e* and the lighter cam shaft *f*, friction gearing being adopted. The front columns carry the guides *g*, being prolonged upwards for that purpose, as well as strong brackets *h*, on which the cam-floor *i* is laid, and to which is also attached the water main *k*. The tappet *l* is of the obsolete threaded pattern, and the feed opening *m* will accommodate anything as large as a man's head. An advantage for transport lies in the fact that the frame can be made in pieces of reasonable size and weight, adapted to bolting together in construction, and that without sensibly introducing an element of weakness.

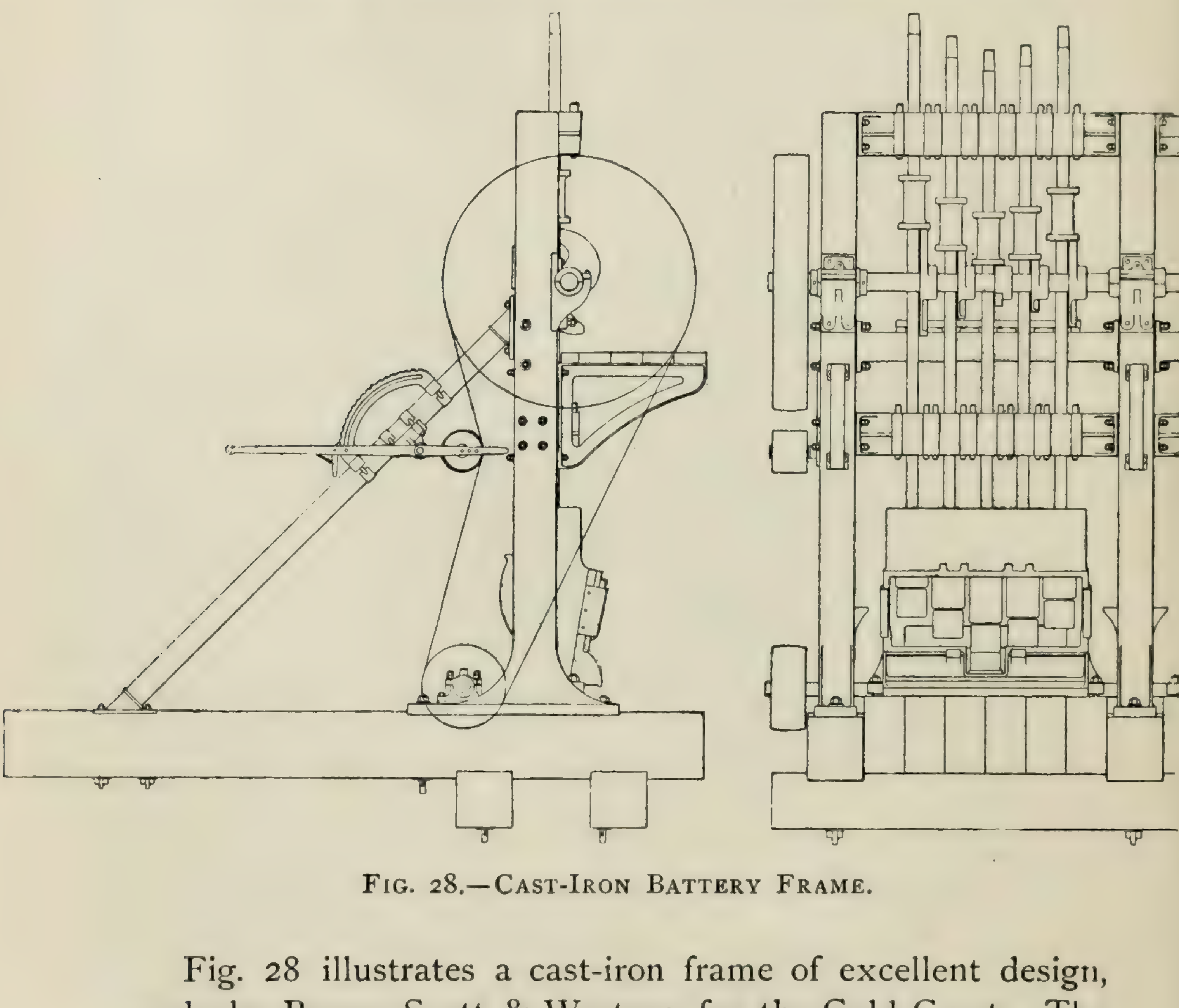


Fig. 28.—Cast-Iron Battery Frame.

Fig. 28 illustrates a cast-iron frame of excellent design, made by Bowes Scott & Western for the Gold Coast. The columns are of hollow box section, and are steadied by tubular steel struts. It possesses but few joints, and being

carefully adjusted and fitted before despatch offers considerable facility in erection where skilled mechanics are not abundant.

Steel.—Wrought iron and more recently steel have been adopted by several English makers for battery frames intended for West Africa and other countries where the white ant prohibits timber, or where the difficulties of transport are great, and wood cannot be got on the spot. In such cases, the

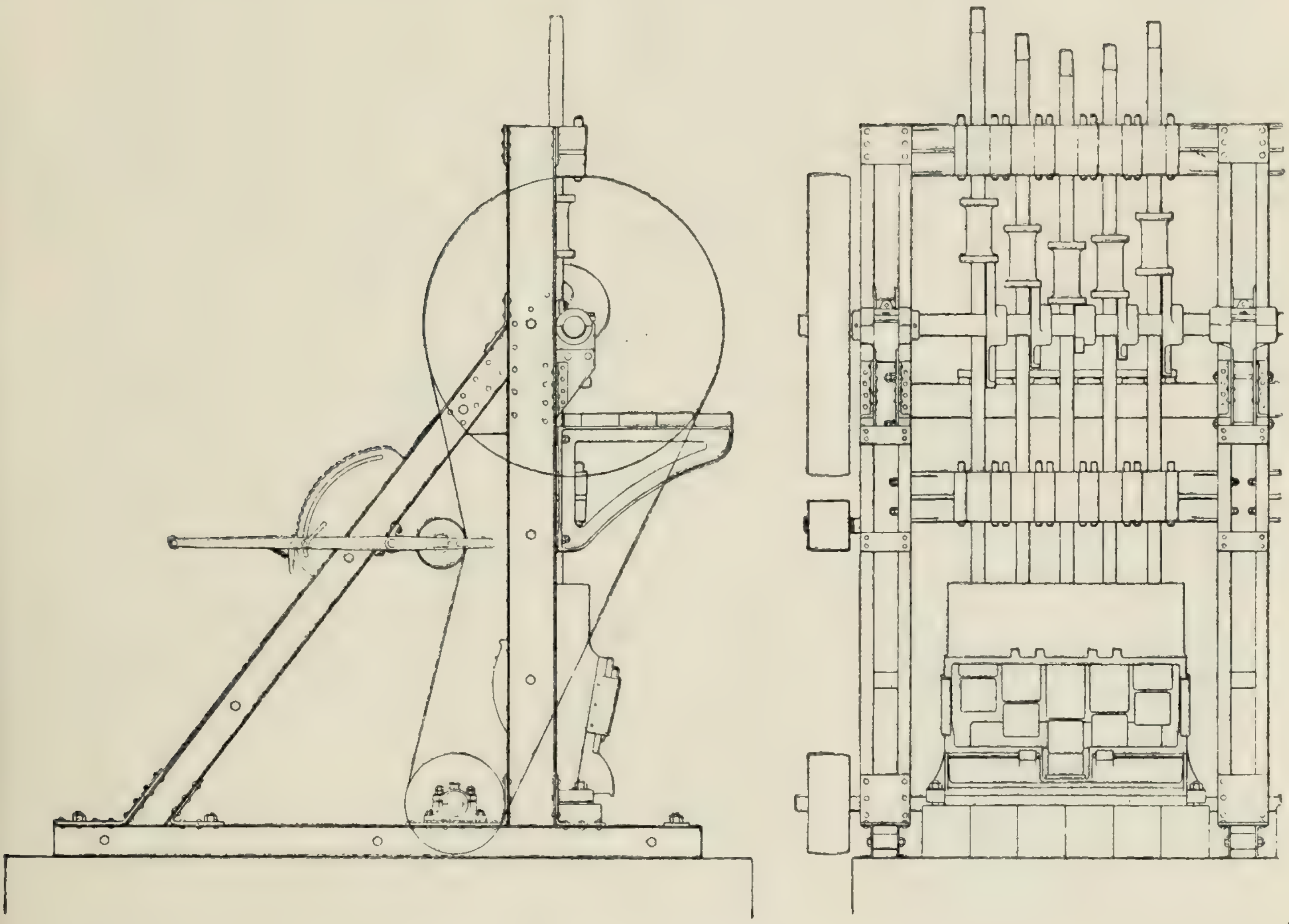


FIG. 29.—STEEL BATTERY FRAME.

foundations are made of concrete or cement masonry, with substantial base bolts built in. Commonly the pattern adopted follows the timber *A* frame, parallel lengths of channel iron bolted together in pairs forming each leg. Similar duplications form the transverse sills, to which both the frame "posts" and the struts are bolted. The upper end of the strut is directly opposite the bracket which carries the cam shaft. Greater strength is secured by the ⋏ pattern, but both back and front of battery are much encumbered. These frames admit of

division into very small and easily transported sections, but they suffer extreme vibration when at work.

An example of a built-up steel girder frame, by Bowes Scott & Western, is shown in Fig. 29.

**Mortar Boxes.**—The mortar box or coffer is that portion of the battery in which the stamps (or pestles, as they might be called) operate to pulverise the material subjected to them. It consists essentially of a cast-iron box or trough, about 4 ft. long, 4 ft. deep and 1 ft. wide inside, preferably and generally cast in one piece, though occasionally made in sections bolting together, when intended for situations where the great weight (2 to 3 t.) of a single casting would prohibit transport. A general idea of the mortar box placed in position on its blocks is shown in Fig. 30. The feed opening *a*, extending practically the whole length of the mortar, and about 3 in. wide, receives the already partially broken ore, either from an automatic feeder or by hand-feeding. The lip of the feed opening is sometimes turned inwards, as shown, in order to minimise the escape of pulp and water splashed up by the stamps; but this provision is often dispensed with in machine-fed batteries, as the lip of the feeder in a great measure serves the same purpose. Care must, however, in that case be taken that the entrance to the feed opening is somewhat less than the rest of the passage, so that no risk may be run of material packing in it. The turned-in lip has a similar effect in practice. On the opposite side to the feed opening is the discharge aperture, furnished

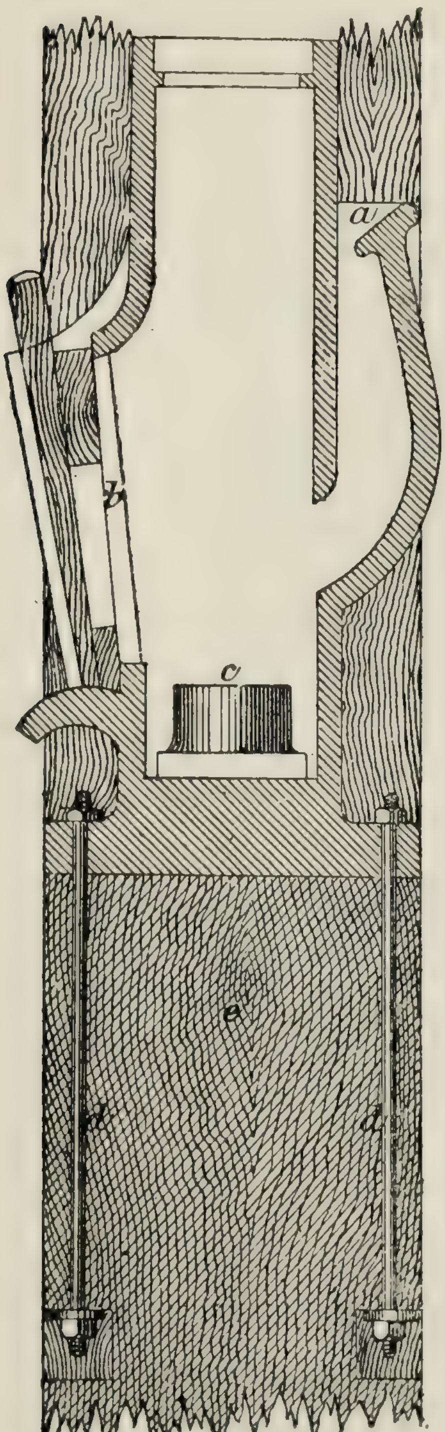


FIG. 30.—MORTAR BOX.

with a screen *b*, through which the stamped material finds its escape. This also extends nearly the full length of the mortar, and is 12 to 18 in. wide, its bottom edge being always above (but at very various distances) the die *c*. The illustration shows the holding down bolts *d* as they are generally fixed in the mortar blocks *e* (see p. 53).

*Material.*—It is hardly necessary to insist that a structure designed to withstand the terrible jar and strain of the continuous pounding of 5 stamps, each weighing in the neighbourhood of half a ton (sometimes more), shall be made of good stuff giving a fine-grained homogeneous texture to the finished casting. Prof. H. Louis * goes so far as to indicate the exact mixture which should be used "in Great Britain," and specifies "good machine scrap, Scotch No. 3 foundry pig, and No. 3 Bessemer pig;" but he omits to state in what proportions. His object, however, is to secure "a fairly high percentage of graphitic carbon, moderately low silicon, and phosphorus not exceeding ·5 %." A very small minority of the mortar boxes in use throughout the world are cast in Great Britain, and, in the case of those few that are, probably no two makers adopt the same formula. At mine foundries where mortar boxes have been cast under the author's direction, a mixture of ⅔ hematite pig and ⅓ good ordinary scrap was used, and answered very well. In fact, it is the rarest thing to see a mortar box destroyed through inherent defects in the material. No doubt a badly cast or badly cooled box will suffer fracture under the strain, but the great majority of all boxes, even those made in small Colonial foundries, wear out either by honest attrition or by carelessness which leads to an injury. It will be obvious to any founder that the great disparity between the thickness of metal in the bottom and of that in the four sides demands particular care in cooling or annealing, so that the shrinkage strains set up may not fall unequally on any part of the box. The mass of metal required to be poured at once places it rather above the capacity of the cupola and ladle to be found at an ordinary mine foundry, and it is as a rule not worth attempting; but there are situations

* 'Handbook of Gold Milling,' p. 128.

when it becomes of some moment to be able to supply one's own wants in this respect, and given a good pattern and core box, a competent moulder, and suitable appliances, there need be no hesitation in tackling the job.

*Dimensions.*—The varying conditions of gold milling in different parts of the world have led to the evolution of many forms of mortar box—all, it may be said, showing but little resemblance to the original prototype, which is now extinct and not worth exhuming. The essential dimensions bearing upon practical work are :—1, thickness of base; 2, length of box inside; 3, width of box inside at level of die and of discharge; 4, distance between bottom of discharge opening and top of die.

The thickness of the base of the box may range from about 4 in. to even 9 in. It bears a direct proportion to the foot-pounds of work done by the battery, its purpose being to support the strains caused thereby, and dependent on the weight, speed, and drop of the stamps. While an excess of metal is only wasteful, it is better to err on that side than on the other. Too light a base will be insufficiently rigid, and though it may not crack from the result of the blows, the efficiency of the battery will be impaired. A really dense and massive anvil is essential, and this is sometimes further provided by a false bottom or die. An average weight of metal in a mortar box is $2\frac{1}{2}$ t., that is for the normal 850 to 900-lb. stamp. For lighter stamps, the weight of the casting may run down to $1\frac{1}{2}$ t., or even less for very light stamps, but they are not in common use. The big stamps favoured in America and South Africa, of 1000 to 1400 lb., require 3 to 4 t.

The internal length of the box is adapted to the size of the face of the shoes and dies. Ordinarily this figure determines the diameter of the base of the dies, and these latter are packed in so closely as only to admit of their handy insertion and removal. But this rule has been broken on the Rand, the arguments there advanced being that, when the dies fit almost exactly into the space at the bottom of the mortar, they are liable to become wedged so tightly as to hinder their ready

withdrawal, and that, when some play is allowed them, it facilitates keeping the stamps truly vertical over them. There does not seem to be much basis to these arguments. In the first place, there are much simpler ways of expediting the removal of the dies, as will be described under that heading; and in the next place, the provision of a certain amount of freedom for the soles of the dies is exceedingly liable to result in their shifting materially before sufficient stamping has been done to pack the pulp round them, so that the efficiency of the stamps is likely to be reduced by some shoes having only a portion of a die beneath them. With proper guides and a solid foundation, the stamps should always hang plumb.

The width of the box at the point where the work is done and the height above the same, as well as the width of the box at the point where the discharge takes place, are governed by the extent to which it is desired to effect amalgamation in the box. The original battery was designed as a crushing machine simply; but the opportunity it affords for at once bringing about a union between the liberated gold particles and the mercury introduced into the box, as well as the amalgamated surfaces attached to its interior, has led to a partial subordination of the crushing duty to the amalgamating duty. On this point more will be said in a future chapter, but it is necessary to state here that the conditions which favour the one operation are a hindrance to the other. When amalgamation in the box is sought, facilities are given by widening the box and heightening the space between die and discharge; when it is not desired, the mortar is made as narrow as possible, and the discharge is lowered. A few years ago, the former condition was almost universal (though not always in the same degree), because of the importance of making the battery extraction as complete as could be, leaving the tailings as poor as possible; but the successful introduction of modern leaching methods, for extracting from tailings the gold which no battery could ever recover, has altered the commercial aspect of the case, and on most fields there is a reversion to the original proposition, viz. regarding the battery as a pulveriser only, and seeking to increase its duty in that capacity

alone. There are, however, many intermediate stages, and quite a variety of mortars are in general use.

*Linings.*—It has come to be recognised that while certain parts of the mortar box are subjected to severe wear and tear, the remainder, forming by far the largest part of it, practically suffers nothing in use, and is as good as new, even when the box is worn out. The greatest attrition takes place on the shoulder or shelf (marked *c* in Figs. 31, 32) which receives the ore as fed into the box ; and on the four sides surrounding the dies, extending for about 9 in. above the top of the die, that is to say as far as the splash of pulp is carried by the falling stamps. Thus it is quite common to find boxes in use exhibiting no signs of general decay, yet wearing patches of sheet iron lined with strips of old vanner-belt or blanket and held on by bolts passing through the shell of the mortar ; and when they have gone beyond this stage, and are thrown out as waste, it will be found that their total weight has not been diminished 5 %. For a very long time it was an accepted condition that mortar boxes must be replaced once in every 4 or 5 years, and in many instances their life was even shorter than that. An effective remedy for this anomaly, which is most obvious, and finds its counterpart in every other machine liable to extreme wear on certain parts, is the introduction of lining plates to cover the exposed spots. These liners can be made of metal having much greater durability and being much more costly than is admissible for employment in the principal casting. Thus cast steel at once suggests itself, and is most generally availed of, while even more lasting material, such as chrome steel and manganese steel, can be employed if the use of bolts in fastening the liners in place is avoided, the latter especially resisting any tool that could be made for drilling holes in it. It is quite an easy matter to cast the mortar box with lugs inside, behind which the liners can slip into place, and if the end plates are made to reach 2 in. or so higher than the side liners, lugs need only be provided for them, the end plates themselves holding the side ones from falling inwards. For this purpose nothing is better than manganese steel, simple clean castings being all

that are required, without any tooling whatever. When worn, they can be taken out and replaced by new, just as easily as dies are exchanged. There is an additional advantage about the use of liners, that an abundant width can be given to the box in the first place, and be afterwards adjusted to the demands of the ore (when ascertained by practical experiment), by using thicker or thinner liners. With a different object, viz. to lessen the depth of discharge, sometimes a false die, consisting of a single slab of metal, is laid in the bottom of the mortar.

*Typical Mortars.*—A number of representative forms of mortar are shown side by side in Figs. 31, 32, so that their differences can be readily appreciated.

In A is seen an ordinary mortar adapted for use where but little amalgamation in the box is intended: the feed opening *a* is contracted, so that anything which enters can readily pass down *b*, and the slope of the lower portion, ending in a shoulder *c*, directs the material fairly on to the die *d*; amalgamation is only partially provided for, by introducing a copper lining plate at *e*, along the lower edge of the discharge *f*. The width inside the mortar at die level is about 12 in., and at discharge level about 15 in., and the depth of discharge when dies are new is about 4 in.

B aims at more thorough amalgamation, and therefore the ore fed in at *a* and passing down *b* falls on a sloping shelf *c*, which shoots it toward the die *d*; but not in the same effective manner as in A, because the lifted shoe will hardly ever clear the end of *c*, and an extra large piece of stone may easily cause damage; *c* would be better placed 2 or 3 in. lower, and might well be of stouter metal, as the wear on it is excessive. Besides the strip of copper plate at *e*, fastened to a wooden block under the screen-frame *f*, a second copper plate is added at the back *g*, being held by bolts passing through the mortar, and protected from the feed by the shelf *c*. The inside width and depth of discharge are virtually the same as in A, but the width at discharge is increased.

C is known as the "Homestake" pattern (of which there have been endless modifications). It has earned a great

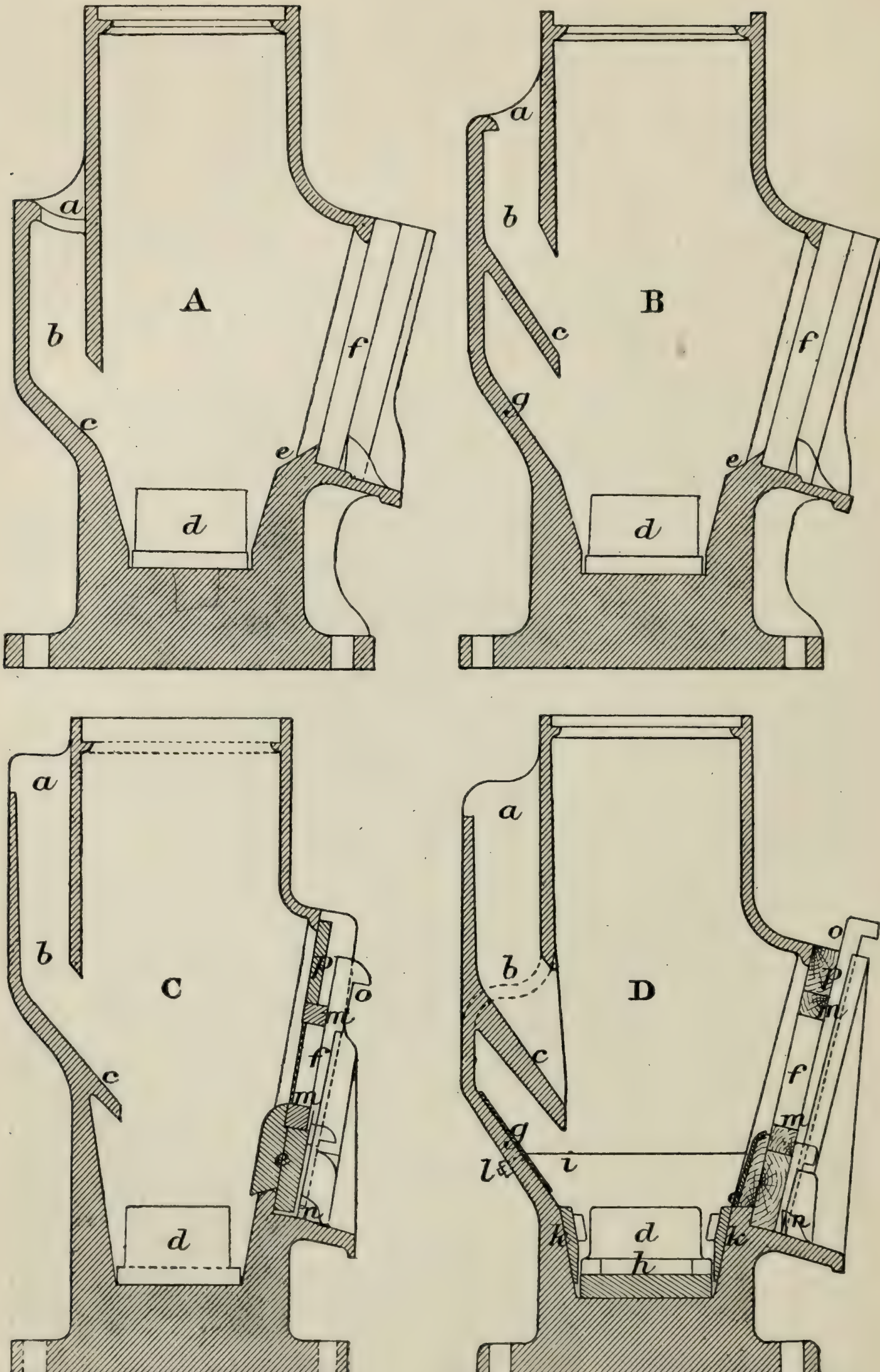



FIG. 31.—MORTAR BOXES.

reputation, and been much copied all over the world—chiefly, it would seem, because the duty of the Homestake batteries is high,—but it possesses glaring faults. Thus the feed opening *a*, with an extreme width of 4½ in., actually contracts at *b*, in opposition to all reason, and the shelf *c*, intended to direct the ore on to the die *d*, really brings it in contact with the rising and falling shoe ; and moreover *c* extends so far towards the centre of the box that the free space for admission of ore is reduced to an absurd minimum : indeed, it is quite a common occurrence for ore to become so wedged as to need forcing out with a heavy bar, a condition of things for which there is no excuse. The inside width at die level is about 13 in., and at discharge level over 18 in., but the shelf *c* tends to drive the wash over to the front and hastens the issue ; the depth of discharge over the lip is very low, not much more than 1½ in., but the insertion of a chuck-block *e*, carrying an amalgamated copper plate, increases it to about 9 in., and reduces the depth of screen *f* to approximately 7 in.

D is an "improved Homestake" mortar. The contraction between *a* and *b* still exists, though in a slightly lesser degree ; *c* is shortened, lowered, and thickened, improving its delivery to the die *d* and prolonging its life. The chuck-block with its copper plate at *e* is made in a series of three sizes, viz. 6 in., 8 in., and 10 in. deep respectively, so that the level of the issue through *f* may in some measure be kept constant, notwithstanding the wearing down of the die *d*. A second copper plate *g*, ¼ in. thick, is provided at the back. The dies *d* rest upon a 2½-in. cast-steel false die *h*. End liners *i* and side liners *k*, of ⅞ in. cast-steel, wedge-shaped at bottom, surround the dies. It will be noticed that only an inch or so separates the points at which the steel liner *k* ends and the copper plate *g* begins, and the utility of that portion of *g* which is below the holding-bolt *l* is more than doubtful, for it must encounter almost as much scour as the upper half of the end liner *i*. The width at die level is about 13 in. and at discharge level about 20 in.

E is the type of mortar favoured in Amador county,

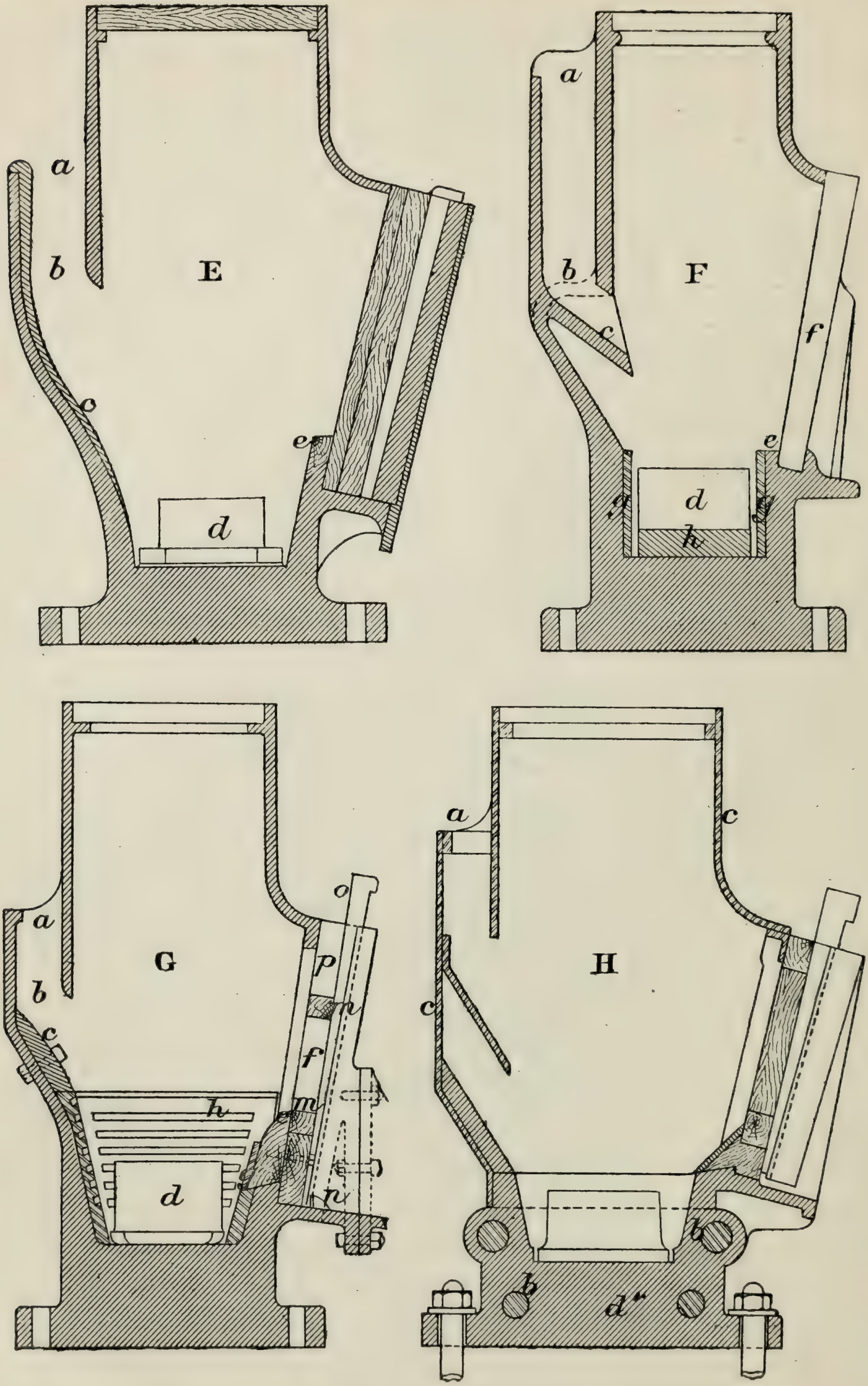



FIG. 32.—MORTAR BOXES.

California, where facilities for inside amalgamation are sought. The feed is well designed, being widened slightly from *a* to *b*, and the shape at *c* serves to deflect the stream of ore on to the die *d*, without itself incurring excessive wear; moreover the latter is taken up by a 1-in. steel lining plate, held in place by bolts through the box. The width at die level is about 14 in. and at issue level 18 in.; here a copper-plated chuck-block *e* is situated. In most respects this is an ideal Californian mortar, with the exception that it is slightly ($\frac{1}{2}$ to $1\frac{1}{2}$ in.) wider than the generality.

In the typical Colorado mortar, internal dimensions reach their maximum. The usual width at level of issue is 24 in., and the depth of discharge between top of die and bottom of screen ranges from 13 to 16 in. according as the dies are new or worn. The crushing duty of these batteries is reduced to a minimum, but little exceeding 1 t. per 24 hours; while the amalgamating capacity, due to added mercury and inside plates, excels all other patterns.

F is from a design introduced into some of the Transvaal mills, where the attrition from crushing banket is found to be excessive. The feed possesses the common fault of contraction between *a* and *b*, exaggerated by the sharp angle at which the shelf *c* sets off. This latter is so flat that it must suffer extreme wear from the falling ore; moreover it protrudes too far into the box, and delivers too high above the die *d*, which rests on a cast-iron false die *h*. The false dies are cast usually in two pieces, wedge-shaped at one end so as to overlap each other in the middle of the box, and sometimes several are used of varying thickness to suit the wear of the dies. There is no attempt at inside amalgamation, and even the customary chuck-block at *e* is dispensed with, so that nothing impedes the rapid discharge through *f*. But $\frac{3}{4}$-in. steel or cast-iron lining plates *g* are dropped in vertically along the front and back of the dies, and sometimes at the ends also. The width at die level is only $11\frac{3}{4}$ in. and at issue level 15 in., while the depth of discharge does not exceed 2 in. In this type, therefore, everything is sacrificed to crushing duty, the aim being not to retain the pulp in the box one moment

after it is sufficiently fine to pass the screen in *f*. It is the exact antithesis of the Colorado mortar.

In G is shown a modern and ingenious pattern which has proved highly successful in the well-known Alaska Treadwell mill, and has in consequence been copied at several of the Rand batteries. It aims at effecting inside amalgamation by the aid of steel liners. The feed is on correct principles, the aperture being less at *a* than at *b*. The steel liner at *c* too presents a great thickness where the wear is most, and its contour is such that with a minimum of resistance it passes the stream of ore precisely where it should go on the die *d*, and makes interference with the rising and falling stamps impossible. The through-bolts which hold it in place can be introduced at the ends, beyond the feed, or may be made to go only partially into the liner *c*; a nut standing up in the path of the ore feed would, of course, be destroyed in no time. At *e* is a chuck-block covered with copper plate, forming the bottom of the discharge *f*. Below *e* is a solid steel liner *g*, but the steel lining plates *h* used at the ends and back of the box are provided with horizontal grooves, affording opportunities for amalgam to collect. There is little doubt that this will take place to some extent, as it is found that some amalgam will usually accumulate on the top edges of ordinary steel liners and around bolt-heads, and so on. But one would anticipate that its collection from these recesses would be a troublesome job, and certainly much more so than its removal from a copper plate. Still, with a free milling ore, it may well repay all trouble. The catchment area is very much greater than copper plates ever present, and there are better opportunities for taking advantage of the specific gravity of gold and amalgam, which is a much larger factor in amalgamation than is generally conceded; while even if they do no good, there is at any rate no scouring away of copper plate, carrying with it some mercury, gold, and silver. For a rapid-issue mortar the dimensions are above the average: the width at die level is about 12 in., and at discharge level fully 20 in., while the depth of discharge with new dies is about 5 in.

The mortar box shown in Fig. 33, made by Bowes Scott & Western, is designed specially with a very large opening where the screens are fitted, so that when they are removed,

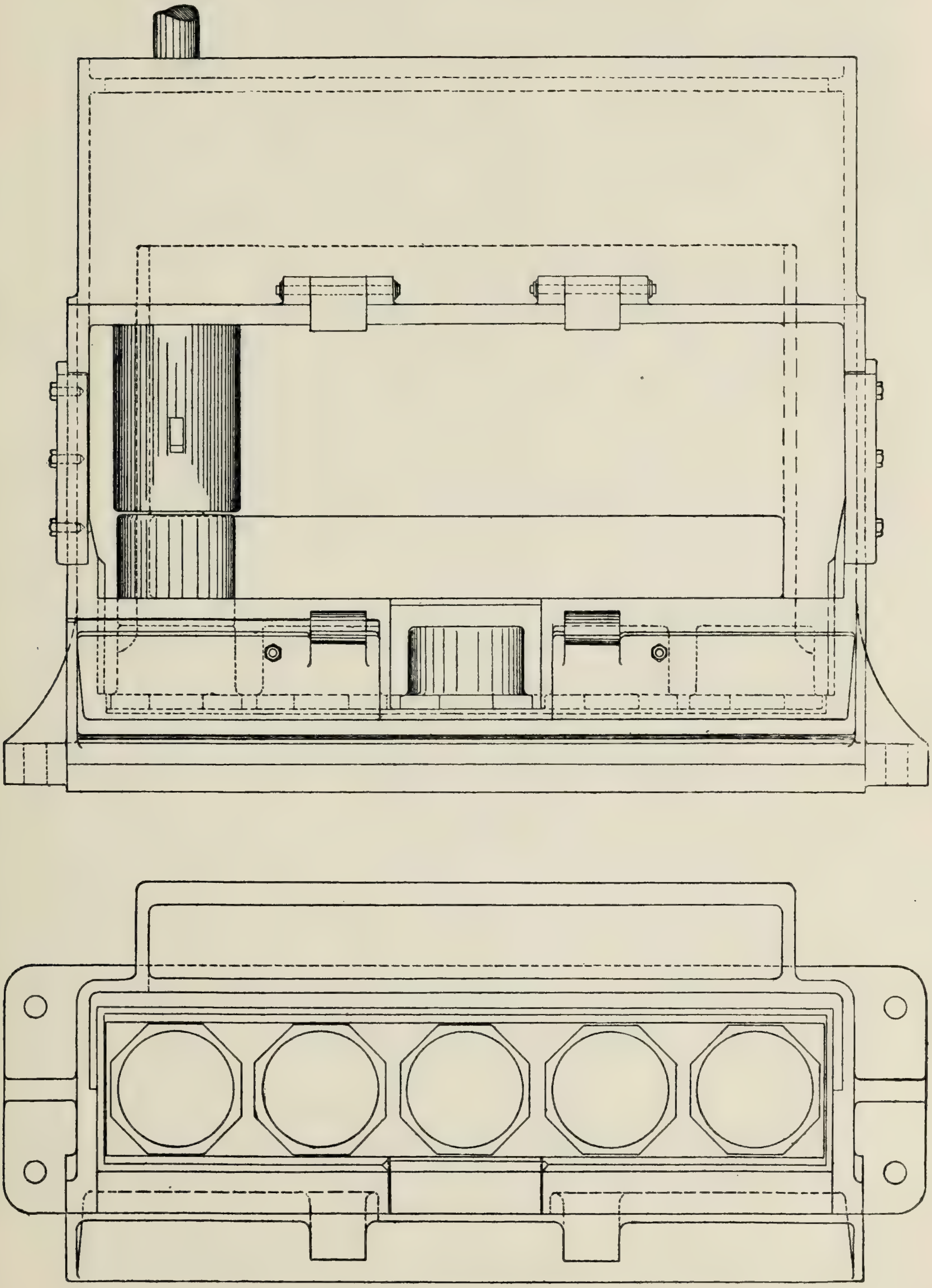


FIG. 33.—MORTAR BOX.

the interior is exceptionally convenient of access. In addition to this, a removable door in the middle, lower down, facilitates replacing the dies, etc. This door is made water and mercury tight by simply turning one screw; but even should there be through carelessness a leakage here, it all has to pass the drop boxes and plates subsequently. The interior of box, where it is subject to excessive wear from the splashing, is armour clad with steel plates. These plates are so constructed as to be easily renewable.

*Sectional Mortar.*—A built-up mortar for mule-back transportation, as made by Fraser & Chalmers, is shown at H, Fig. 32. The base *a* is of cast-iron, in 11 sections, planed and fitted to each other, and bound together longitudinally by perfectly fitted bolts *b*. The housing *c* is of boiler-iron, to reduce weight and risk of breakage, and the joint is made in a way to secure accurate fit as far as that is possible. In setting them up, it is necessary first of all to scrupulously clean all planed faces, and file down any inequalities caused by rough handling. Bolts and bolt-holes must be cleaned and oiled. Planed faces should receive a thin coat of paint when putting together, and be held in place by temporary bolts until exactly lined, avoiding the insertion of two bolts on the same side. When true, a pair (diagonally *b b*) of the proper bolts may be entered, using a wooden block between the bolt head and the driving hammer, and screwed up tight; the temporary bolts are then withdrawn and replaced by the remaining pair of true bolts. In adding the housing *c*, extra strips of copper are introduced between the cast-iron and the boiler-iron, and the latter is hammered all round as the screws are tightened up. The copper serves as a sort of caulking, but actual water-tightness is very difficult to attain. When finished, some weak acid, or better still a decoction of wood ashes, may be used to wash out all traces of grease. The entire box weighs 5000 lb., and no piece exceeds 300 lb.; but two pieces of 300 lb. each (they hang beside the animal, and must counterbalance) make no small load for any animal over bad roads—and they must be bad for a sectional mortar to be required!

*Multiple Discharge.*—It has long been regarded as an axiom in stamp milling that the capacity or duty of the stamp is governed by the freedom of issue provided, or in other words that the screen area controls the output. There are not wanting authoritative assertions that the increase of capacity beyond that secured with the ordinary single discharge should theoretically be almost in direct ratio to the additional screen area. Apparently this theory has been deduced from certain experiments made years ago—under what conditions of control is not clear—tending to show that the time required by crushed pulp (which had already been through a battery) to issue through the screens when again fed into the battery equalled the time occupied by a similar weight of rock which had not been previously crushed. This was regarded as proof positive that the restricted screen area furnished by the usual front discharge limited the effective work of the battery, and the natural outcome was an adoption of mortar boxes arranged to accommodate back and end screens as well as front ones. Great ingenuity has been displayed by numerous inventors in designing such boxes, and the flood of invention is still flowing in the same channel, new forms of mortar affording the maximum of screen room, by placing a single stamp in the box and surrounding it by gratings, appearing in the most recent patent records.

Unfortunately the whole idea is based on a misconception. No stamp battery (excluding steam stamps, which will be dealt with later) has yet been made which can crush up to the maximum capacity of the single screen. This has been very clearly shown by D. B. Morison and D. A. Bremner,* in a set of experiments conducted on thoroughly scientific lines, not with ordinary gold-bearing quartz, it is true, but with material which may be regarded as equally conclusive. An ordinary cam-lifted gravitation stamp (single) was used in a Thompson "unit" mortar box with all-round discharge, and 5 Morison "high speed" stamps, each weighing 1428 lb., were employed in a double-discharge mortar of ordinary pattern. The materials were in each case double-screened to 1½ in. gauge, and

* Trans. Inst. M. & M., viii. 166 (1900).

contained no fines ; they were accurately weighed, and automatically fed, the load on the dies and the depth of discharge being kept as constant as possible.

The "unit" mill used screens of 500 holes per sq. in., a 4½-in. discharge, and a water ratio of 10 to 1. Operating on whinstone, it had a duty of 3·36 t. with one screen and 3·41 t. with three screens ; operating on quartzite, the figures were 6·76 t. for one, 7·02 t. for two, and 6·75 t. for three screens; and operating on broken firebrick, which was used to obtain a high rate for the purpose of ascertaining, if possible, the point at which the screen would control the crushing, the outputs were 8·37 t. for one, 8·27 t. for two, and 8·52 t. for three screens. A duty of 6·76 t. per 24 hours, on an effective screen area of 228 sq. in. (19 by 12 in.), is equivalent to 59·29 lb. per effective sq. in. per 24 hours; and a duty of 8·37 t. (which could never be attained on ordinary quartz) means 73·42 lb.

The "high speed" mill used a 3-in. discharge and a water ratio of 10 to 1, and operated on quartzite only, that giving the nearest approach to quartz-milling conditions attainable in England. The actual front screen area was 885 sq. in. (17 by 52 in.; and the effective area, 624 sq. in. (12 by 52). With a mesh of 700 holes per sq. in., the duty was 7·27 t. for one screen and 7·45 t. for two; at 500 holes, it was 8·35 t. for one and 8·24 t. for two; and at 324 holes, 9·05 t. for one and 8·95 t. for two. At 900 mesh (much the same as No. 7 needle, and a common gauge in gold mills), the duty was 7·2 t., or 115·38 lb. per effective sq. in. per 24 hours ; at 700 mesh, it was 7·27 t. or 116·5 lb. per sq. in.; and at 196 mesh, 9·81 t., or 157·21 lb. The last named is much coarser than is usual in batteries, being only 14 holes per linear inch.

These experiments clearly show that the screen area of an ordinary 5-stamp front-discharge mortar box is capable of passing the output of a battery having the duty of 9·81 t. ; so that in ordinary practice it is not worked up to half its capacity, and there is no need to seek its amplification.

A striking confirmation of this important fact came under the writer's own observation in actual work. At the Aladdin's Lamp Co.'s battery, Lucknow, New South Wales, he found a

Krupp ball mill used for wet crushing quartz, installed beside a 10-stamp battery. This mill had a larger output than the 10 stamps, reducing to the same gauge. It being impossible to effect any amalgamation in the ball mill, its pulp was run into the nearer of the two boxes of the stamp battery, the screen area and mesh in both boxes being identical. Yet there was practically no difference between the duty of the stamps in the two mortars. This was not by any means an experiment: it was a steady working condition lasting over many months—until, in fact, the collapse of the ball mill (which was never intended for such a purpose) gave the author his long-desired excuse for dismantling the whole plant, and proceeding on more rational lines.

Having shown that the claim for a multiple discharge is invalid, it may perhaps be regarded as sufficient condemnation of the practice, as no justification can be urged for adding to the cost of the mortar box (as well as cost of running) without some specific gain. On the contrary, the addition of a rear screen brings actual drawbacks with it. In the first place, the feed is interfered with, whether by hand or machine. Then the screen itself is subject to much greater wear and tear, encountering violent blows and consequent damage from comparatively large pieces of rock thrown back by the falling stamps. Moreover it is out of sight and reach: an injury to it cannot readily be seen, and its repair or removal is an awkward matter; consequently double-discharge mortars may be seen in which the back screen has been blocked up. But inside amalgamation will not necessarily be prejudiced, as is supposed by some, because the output is not increased, and an amalgamated chuck-block may take the place of the supplanted inside back-plate. Against end discharges there are not the same objections to be urged, though they do no actual good. It has been said in their favour that "they enable the use of copper tables of an unusual width" (Rickard); but even if so, assuming that they hasten the discharge, they diminish the opportunity for inside amalgamation, and thus undo as much as they do, besides which the width of the

copper tables is certainly not controlled by the length of the front screen.

There is hardly an example of back or end discharge mortars in current use, with the exception of some instances in Australia. Rickard * describes two in Victoria. In both, the screens stand vertical, as they do in most batteries built by Australians. That at Clunes has front and back issue, and operates on stone of unusual simplicity; the depth of issue and gauge of screen are alike for both discharges. That at the Harrietville mill has a most unconventional arrangement: a straight section in front, reaching from opposite the middle of No. 2 die to the middle of No. 4; and a curved section at each end facing about half the circumference of No. 1 and No. 5 dies respectively. The front screen has 240 holes per sq. in., and the end ones 175; each screen is 2 ft. long. Although of coarser mesh, the end screens wear out first. The arrangement gives the impression of needless complication without any corresponding benefit.

**Screens.**—From what has already been said, it will be understood that the discharge apertures in the mortar box are fitted with screens or gratings, through which the issuing pulp must pass. These consist either of sheet metal in which the requisite holes have been punched, or of netting woven of fine wire. In either event, a special frame is necessary in which these comparatively fragile articles can be secured.

*Frames.*—As can be seen from the various illustrations of mortar boxes (Figs. 31 and 32), the reception of the screen frame is provided for in casting them. All the examples given show the screen leaning outwards at an angle of about 10°, because this is the established practice on all fields where gold milling is conducted on modern lines. Most Australian mills adhere to vertical screens, for no reason except an ignorant copying of early forms. The inclined screen possesses two distinct advantages: (*a*) that its upper part is farther removed from the falling stamp, is therefore less liable to injury from coarse rock being dashed against it, and conse-

* T. A. Rickard, 'Stamp Milling of Gold Ores,' pp. 108, 157, 160–2.

quently lasts longer; (*b*) that opportunity for escape of pulp is materially increased, so long as the degree of slope is not so great as to make it inaccessible to any but the finest and most buoyant particles.

Besides the recesses which are cast in the cheeks of the discharge aperture, and which form a seating for the screen-frame, lugs are also cast on the lip of the mortar, rising about an inch above the general surface. The holding faces of all these should be made free from the inequalities and roughness of casting, as wedges have to be driven against them. The full depth of the discharge aperture is not occupied by the screen-frame. Below is usually a chuck-block, on which it rests ; and above is a space which is often filled with a simple board, sliding down in the same recesses as the screen-frame and bearing on it, and perhaps as often covered only in a casual fashion by a loose strip of sacking or canvas. Abundance of room in the discharge aperture is most desirable for convenience in cleaning up, and replacing worn-out shoes and dies. In Fig. 31, examples C, D, G show details : *e*, chuck-block ; *m*, screen-frame ; *n*, lugs ; *o*, steel wedge-keys ; *p*, board or space above frame.

Screen-frames are made both of iron and of wood. The former are general in Australia, where, indeed, iron seems to be in special favour with mill builders, perhaps because the country produces so little wood fit for joinery purposes. Outside Australia, wooden frames are almost universal. Iron frames are heavy, and it is impossible to make such a close joint with them as with wooden ones ; they are often divided into 2 or 3 compartments by strengthening-bands crossing them at regular intervals. This, of course, reduces the available screen area, though that would not seem to be a matter of much importance, as the capacity of the screen is never nearly reached. Apparently these supports have been introduced with a view of prolonging the life of the screens rather than that of the frames. Wooden frames are made of 3 in. by $1\frac{1}{2}$-in. laths, securely mortised at the corners, and lined, on the side opposite to where the screen is held by small tacks, with a strip of 1 by $\frac{1}{8}$-in. hoop-iron screwed on to each end : this

takes the wear of the wedging keys *p*. When inserting a frame in the seating, strips of wet blanket equal in width to the frame are laid all around it on the screen side, that is facing inwards to the box. These make a joint which will prevent the escape of any pulp not fine enough to pass the screen, but a really tight joint is not easily got. After a screen-frame is in position, it is held by driving down the forged steel wedging-keys *p*, and putting small wedges between the frame and the lugs *o* on the mortar lip. These last are sometimes replaced by straps, cotter bolts, and other contrivances, but anything in the nature of a thread is soon ruined by the attrition of the discharging pulp if exposed to it.

As the wear of screens is very unequal, and there is a constant liability to sudden damage apart from actual wear, spare sets should always be ready for insertion at any moment. Total cessation of work in that particular battery is entailed so soon as a screen breaks down, and continues until it is replaced, so that the time occupied by this operation must be reduced to a minimum. A careful millman will replace screens *before* they break or cease to do efficient work, and the man who should hang up his battery now and again to look for and replace broken screens, as one authority states, would soon be seeking another job if there was any proper management. The working face of a screen is seldom more than about 12 in. wide, and only the lower half of that really encounters much wear, so that usually a screen will bear turning upside down when partially used.

*Punched sheet screens.*—These are made of Russia iron, cold rolled homogeneous steel, tinned iron, and copper, while certain alloys have been tried. The two first named have by far the widest use, the sheets varying in thickness from 28 B.W.G. (·014 in.) to 22⅝ B.W.G. (·0269 in.), and in weight from ⅔ to 1¼ lb. per sq. ft., the most commonly employed being about 24 B.W.G. and 1 lb. per sq. ft. Tinned iron is preferred in some places, as being thinner, and thus offering less impediment to the discharge ; the tin is first removed by burning. Copper is confined to Australian batteries, and can be applied only where no amalgamating is done in the mortar

box. An aluminium bronze (95 % Cu, 5 % Al) was tried at the Homestake mills, South Dakota, proving very durable, and not liable to amalgamate; but though the worn screens had a market value, and thus diminished the nett price of the material, it was abandoned as too costly.

The perforations in punched screens are of two kinds—

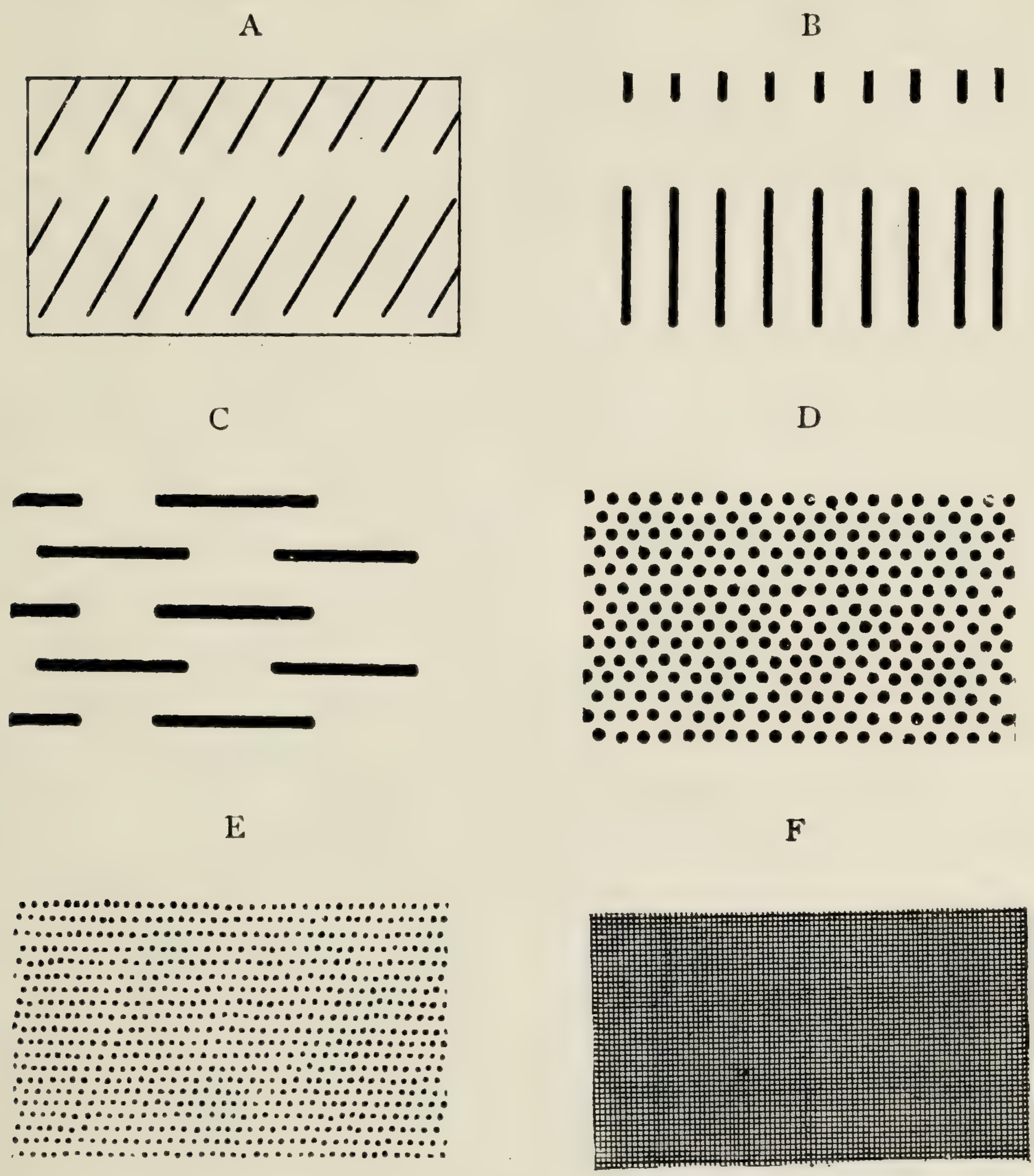



FIG. 34.—SCREENS OR GRATINGS.

plain and "burred;" in the former, the hole is made by cutting a piece of metal clean out; in the latter, the punch forces the metal away from in front of it and curls it back around the hole. Each class has its advocates. The plain puncture offers readier escape for the pulp, but sooner wears to an increased

size. The burred pattern hinders the discharge slightly till worn down, as the burr is invariably turned towards the issuing stream of pulp; but the expanding outlet makes it much less liable to choke, anything that enters being able to escape. The shape, disposition, and size of the orifices in punctured screens are subject to the greatest variety. When the holes are round, the gauge is known by the number of perforations to the square inch ; when they are slots, they are called after the " number " of the needle that will pass through them, and a corresponding " mesh " in holes per linear inch has been somewhat arbitrarily established. As will be seen by a glance at Fig. 34, which shows some typical forms, the actual amount of screen area per square inch is as erratic as the arrangement of the apertures. Even with simple round holes, there is no rule for the ratio of orifice to solid metal, so that the number of holes per inch is really no guide to the size of the holes. Slots may be disposed at any angle, and may break joint, or run in parallel rows. The subjoined table gives the main features of slot screens :—

SCREENS, SLOTTED.

| No. | Mesh. | Width of Slot. | Thickness: | | | Weight per Sq. Ft. |
|---|---|---|---|---|---|---|
| | | | Russian Gauge. | Birmingham Gauge. | In. | |
| | | in. | | | | lb. |
| 1 | 12 | ·058 | 15 | 22⅜ | ·0269 | 1·24 |
| 2 | 14 | ·049 | 15 | 22⅜ | ·0269 | 1·24 |
| 3 | 16 | ·042 | 15 | 22⅜ | ·0269 | 1·24 |
| 4 | 18 | ·035 | 15 | 22⅜ | ·0269 | 1·24 |
| 5 | 20 | ·029 | 14 | 23¼ | ·02425 | 1·15 |
| 6 | 25 | ·027 | 13 | 24 | ·022 | 1·08 |
| 7 | 30 | ·024 | 12 | 24½ | ·021 | ·987 |
| 8 | 35 | ·022 | 11 | 25 | ·020 | ·918 |
| 9 | 40 | ·020 | 10 | 26 | ·018 | ·827 |
| 10 | 50 | 018 | 9 | 27 | ·016 | ·735 |
| 11 | 55 | ·0165 | 8 | 28 | ·014 | ·666 |
| 12 | 60 | ·015 | 8 | 28 | ·014 | ·666 |

The length of slot ranges from $\frac{3}{8}$ to $\frac{5}{8}$ in., and is most commonly $\frac{1}{2}$ in. In Fig. 34, A is a number 9 needle (40 mesh) angle slot; B, a vertical slot; C, a horizontal slot, breaking joint; D has 200 holes per sq. in.; and E has 575 holes per sq. in.

The hard work performed by the screen renders it necessary that good metal shall be used in its manufacture, with perfect freedom from rust and flaws. Russia iron, being very malleable, and having a smooth finish, is perhaps the most durable in withstanding attrition; its approximate cost is 13*s.* to 16*s.* per screen. But in some places an article of local manufacture is substituted. Thus an imitation called "planished" iron, made out of 24-gauge sheets, punched with horizontal alternate slots, ranging from 40 to 60 mesh, is much favoured in Gilpin County, Colorado; the cost seems to be as high as the genuine Russia iron, and the quality is apparently by no means inferior. The term Russia iron is either much abused, or there are vast differences in quality. Thus New Zealand mills claim to have an article, imported from Swansea, costing only 2*s.* 10*d.* per screen. In New South Wales, the author imported from San Francisco, and paid 13*s.* 6*d.* per screen. The Swansea screens endure for only 5 to a maximum of 12 days, while the San Francisco article is always good for 36 days; the duty of the former ranging from 47 to 134 t., as against 300 t. Many of the Victorian batteries employ also a local iron or steel sheet; and in California, tin plate is in general use. The slot-punched hole is as universal in America as the round-punched hole is in Australia and New Zealand. The copper screens used at Clunes, Victoria, as formerly imported from England, weighed $1\frac{1}{2}$ lb. per sq. ft. and were $\frac{1}{16}$ in. thick; but a locally made sheet is now used, and is apparently much thinner, costing 15*s.* 4*d.* per screen. Copper may be adopted with advantage when (no amalgamation being effected in the box) the milling water carries sulphuric acid or sulphates in solution.

The annexed table gives details of punched screens in use on many fields, in a form admitting of ready comparison :—

SCREENS, PUNCHED SHEET.

| — | Kind. | Mesh. | Depth of Issue. | Life. | Duty. | Price. | Cost per ton. |
|---|---|---|---|---|---|---|---|
| California : | | | in. | days | t. | s. d. | d. |
| Amador County . | Russia, slot | 30 | 7 to 8 | 33 | 396 | 16 3 | ·49 |
| Grass Valley . . | Tinplate, slot | 40 | 2 to 6 | 14 to 24 | 119 to 192 | 2 4 | ·15 to ·23 |
| North Star . . . | Tinplate, round | 110 | 4 | 30 | 240 | 2 1 | ·10 |
| Colorado : | | | | | | | |
| Gilpin County . . | Planished iron, horizontal burr slot | 50 | 11 to 13 and 13 to 16 | 16 to 81 | 74 to 460 | 16 3 | ·42 to 2·63 |
| Dakota : | | | | | | | |
| Homestake . . | Russia, plain slot | 30 | 9 to 11 | 7 | 140 | 13 6 | 1·15 |
| New South Wales : | | | | | | | |
| Lucknow . . . | Russia, burr slot | 30 | 3 to 4 | 36 | 300 | 13 6 | ·54 |
| New Zealand : | | | | | | | |
| Otago . . . . | Russia, round | 180 to 200 | 2¼ to 4½ | 8 to 12 | 54 to 134 | 2 10 | ·25 to ·63 |
| Thames . . . | Do. | 148 to 180 | 0 to 5 | 5 to 6 | 47 to 101 | 2 10 | ·33 to ·72 |
| Queensland : | | | | | | | |
| Charters Towers . | Do. | 225 | 0 to 4 | 5 | 70 | 2 10 | ·48 |
| Victoria : | | | | | | | |
| Ballarat . . . | Iron, round | 120 to 200 | 2 to 4 | 6 to 3 | 60 to 134 | ?2 10 | ·26 to ·56 |
| Bendigo . . . | Russia, round | 115 to 170 | 2 to 6 | 10 to 17 | 123 to 190 | ?2 10 | ·18 to ·28 |
| ,, . . . | Local iron, round | 143 | 2 to 6 | 8 | 94 | ?2 10 | ·36 |
| Clunes . . . . | Copper, round | 81 to 100 | 4 to 5 | 25 to 30 | 350 to 403 | 15 4 | ·45 to ·53 |
| Ovens . . . . | Russia, round | 200 to 250 | 1 to 3 | 17 to 20 | 124 to 181 | ?2 10 | ·19 to ·27 |

*Woven Wire Screens.*—Iron, steel, and brass wire are all in use for making woven screens. The mesh is nearly always square, and therefore uniform in a sense; but the gauge of the wire employed is subject to great variation, so that the number of holes per square inch is not a guide to the actual area of aperture. The most important field where woven screens are adopted is the Rand. There, steel wire is almost universal,

and the mesh commonly ranges from 24 to 30, or 600 to 900 holes per sq. in., in which the total area of aperture is 40 to 50 % of the whole screen, each hole having an area of ·0004 to ·0007 sq. in. (Hatch.) Brass wire is used to some extent in America, usually costing about 6*s.* 6*d.* a screen. An example of 60 mesh (3600 holes per sq. in.) is shown at F, Fig. 34 (p. 85).

Wire screening is always sold in rolls, and is cut up for use as required. Much waste is often incurred. When of iron or steel, it is very prone to rust, if the dressing applied by the makers is rubbed off; hence great care must be observed in storing and handling. The dressing is easily removed by a strong heat just before use.

Recent quotations by N. Greening and Sons, Warrington, are as follows:—

Steel wire, light quality:
600 mesh, 2¼*d.*; 900, 2¼*d.*; 1200, 2½*d.* per sq. ft.

Ditto, treble extra heavy:
700 mesh, 5*d.*; 900, 5*d.*; 1200, 6*d.* per sq. ft.

Brass wire:
No. 8 (22), 6*d.*; 10 (24), 5*d.*; 16 (29), 3¾*d.* per sq. ft.

Galvanised wire:
No. 6 (18), 4¾*d.*; 7 (20), 3*d.*; 8 (20), 3½*d.*; 10 (23), 2¾*d.*; 20 (26), 3½*d.*; 30 (31), 4*d.* per sq. ft.

Some interesting particulars of woven wire screens are given in the annexed table:—

SCREENS, WOVEN WIRE.

| — | Kind. | Mesh. | Depth of Issue. | Life. | Duty. | Price. | | Cost per ton. |
|---|---|---|---|---|---|---|---|---|
| California: | | | in. | days | t. | *s.* | *d.* | *d.* |
| Amador County . | Brass | 20 to 30 | 7 to 8 | 25 to 30 | 375 | 6 | 6 | ·21 |
| Grass Valley . . | Do. | 40 | 2 to 6 | 11 | 110 | 6 | 6 | ·71 |
| New South Wales: | | | | | | | | |
| Lucknow . . . | Do. | 30 | 3 to 4 | 6 | 50 | 6 | 0 | 1·44 |
| New Zealand: | | | | | | | | |
| Otago . . . . | Steel | 18 | 2½ to 4½ | 6 | 47 | 2 | 2 | ·55 |
| Transvaal: | Do. | 22 to 30 | .. | 2 to 3 | .. | .. | | .. |
| Geldenhuis . . . | Do. | 24 to 30 | .. | .. | .. | .. | | ·43 |

*Effect of Screens.*—The purpose of the screen is to exert a control over the size of the particles of ore issuing from the battery. This it does in one sense only, viz. it determines the maximum size of the particles, and prevents the escape of anything greater than the particular mesh or gauge adopted. That is an important office, and the mesh chosen is that which experiment (or more often guesswork) has shown to be the maximum compatible with the best extraction of gold contents. In this respect, the screen is the sole determining factor. But it is of equal importance that the degree of fineness shall not exceed a certain limit, and in that capacity the screen is only one of several factors, the most influential being the depth of discharge or issue, and the width of the box at that level. It may be explained in passing that the "best extraction" does not necessarily mean the highest which can be attained, but rather the most desirable from a commercial standpoint; the greatest profit is not often to be had from the maximum extraction. What the ideal figure shall be in any case, and what means shall be adopted to secure it, are matters demanding all the care and thought of the most experienced and scientifically trained engineers. Suffice it here to say in general terms, as bearing simply upon the screen question, that the point to which the pulverisation may be best carried will depend chiefly upon (*a*) the coarseness or fineness of the gold, (*b*) the existence or absence of auriferous sulphides apart from the free gold amenable to amalgamation. Clearly it should never exceed the point at which the best extraction is obtained, firstly because the cost would be added to and the output reduced, and secondly because undesirable complications would be introduced into subsequent processes. But often, that which is advantageous to one stage of the extraction is detrimental to another. Thus the free gold may be so fine that extreme crushing is necessary to liberate it for attack by the mercury used in amalgamation, while that same crushing will result in "slimes" which are difficult to deal with afterwards; and it is particularly necessary to remember that auriferous sulphides are always more brittle, and therefore more prone to

slime, than ordinary quartz. The product of no two mines is alike; and even where there are several reefs or ore bodies in a single mine, they will frequently manifest distinct characteristics. Hence it comes about that the practice of crushing varies from a very coarse to a very fine mesh. At some mills, provision is made for variations in the milling dirt—and this is essential at least in "custom" batteries (which crush for the public)—by having screens of different gauges and types in readiness.

The great desideratum in a screen, once the most suitable normal mesh has been decided on, is regularity of product. This is governed by two conditions: the wear of the apertures in the screen, and the fluctuation of the character of the ore. The former is very commonly ignored in the anxiety to economise screens; instead of their being condemned and replaced so soon as they cease to maintain a proper control, they are allowed to remain until a breakage compels their removal. Having in view the merely trifling cost of screens, approximating only to 1 or 2 % of the total cost of milling, and not exceeding in value $\frac{1}{4}$ gr. of gold per ton (see tables, pp. 88, 89), it is obvious that a very fractional diminution in the ratio of extraction, due to increased size of ore particles passing the screens, will easily counterbalance the small economy that can be gained by using screens up till the last moment of their endurance. At a few mills this is recognised, and the full life of the screen is never attempted; but in the great majority of cases no thought is given to it, and screens are patched and tinkered up to the last extremity. There are, however, instances where the gangue or country accompanying the milling dirt is so much harder than it, that the extra large particles are negligible in value; thus at Lucknow, N.S.W., the coarsest tailings, forming 5 % or more of the whole, assay only "traces."

It has been urged that, in the important matter of uniformity of output, the woven wire screen excels the punched metal, because the wear of the former is more gradual and distributed. On the other hand, it may be pointed out that a piece of sharp rock striking the screen (as will happen) is much

more likely to sever or spread a wire than to cause a fracture in a punched sheet.

Sizing tests on pulp will show that a very large proportion is reduced much below the mesh of the screen. The actual degree will depend on the conditions governing the discharge, and will vary with them. Thus, under usual Californian milling practice, half the pulp passing a 40-mesh screen can also pass a 100-mesh; and in Colorado, almost three-fourths will do so. An instance has been recorded in which fully a third of the ore escaping by a 16-mesh screen (250 holes per sq. in.) went through a 65-mesh (4000 holes per sq. in.). The average work of batteries all the world over is said to result in 25 % to 75 % of slimes, but the latter figure must surely be an exaggeration. Probably about 30 % is pretty near the mark, as a rule, judging by the relative sands and slimes plants erected for treating tailings. At Lucknow, N.S.W., they amount to about 60 %, but the conditions are unique—a most friable ore (mispickel) and soft gangue (calcite) accompanied by extremely hard country (diorite). Sizing tests and corresponding assays are most instructive for indicating what gauge of screen (and the accompanying conditions, of course) will give the maximum product of the most desirable size.

The present tendency is towards coarser milling and an increased duty for the stamps, whereby the cost of milling is lowered. This has been brought about by the introduction of modern leaching processes, which are able to attack and extract gold whilst in a condition of partial if not complete envelopment in other bodies. When amalgamation was practically the only means of recovering gold from ore of ordinary grade—that is excluding the rich sulphides—an attempt had to be made to carry the pulverisation to a point approximating the size of the gold particles themselves, so that sliming was in many cases a necessity. With subtler (than mercury) means of reaching the infinitesimal gold particles, it has come to be generally recognised that the only proper sphere of amalgamation is the coarse gold, leaving the rest to wet methods of extraction, in which slimes are not desired. Indeed, it is not improbable that in the near future, pulverisa-

tion will be conducted in two stages instead of in one as at present: the first crushing sufficing to liberate the more friable and valuable portion of the milling dirt before it becomes slimed, and the harder and coarser but less valuable portion being separated automatically by hydraulic sizers and alone subjected to the additional pounding necessary for its due reduction, if it be worth the operation.

*Life of Screens.*—Quite a number of conditions go to determine how long a screen will last in wear.

Thickness and quality of metal are prime factors in the equation. Thickness is limited by the consideration that freedom of issue is governed by it to some extent, so that a thick punched screen will not have as great a duty as thin; in other words the output of pulp through it per 24 hours will be less. Hence the preference for tinned iron over Russia iron, life being sacrificed for greater duty. In wire screens, not only the quality of metal used, but also the way in which the weaving is done has material bearing. The interlacing should show both warp and weft wires bent at the point of crossing, otherwise there is nothing to prevent a disturbance of their relations and a destruction of that equidistance which is regarded as giving the wire screen a superiority over punched sheet.

The character of the ore exerts an influence, some stone having much more abrasive effect than other. Rand banket has a bad name in this respect, and that is just one of those cases where the barren portion of the milling dirt is the most difficult to pulverise.

The situation of the screen in relation to the scene of pulverising action is the most telling feature under ordinary circumstances. With a narrow box and shallow discharge, no sort of screen can endure very long, 1 to 2 weeks being the limit; whereas with roomy mortars and a deep discharge, 3 to 6 weeks' life is common.

When mercury is used in the mortar box, copper screens cannot be employed at all, as they would soon become amalgamated, and their apertures more or less choked. Brass wire to a less extent is affected also, the action of the mercury

being to render the brass extremely brittle, so that it breaks away before being worn out.

The presence of free sulphuric acid or soluble sulphates in the battery water, whether due to the compulsory use of mine water for that purpose, or to decomposing sulphides in the ore, results in rapid corrosion of iron or steel screens. The effect will, of course, be increased when several mills, situated one below another on the same stream, are drawing their water-supply from the one source and fouling it in turn. Thus at Black Hawk, Colorado, in 5 consecutive batteries the screen-life is on a descending scale, ranging from 81 to 16 days (Rickard). Probably in no other country than the United States would such pollution of running water be allowed ; and it seems remarkable that this condition of things should still survive when the addition—at a trifling cost—of a little lime to the ore would not only remedy the abnormal destruction of the screens, but would also diminish the sulphating and loss of mercury which must occur, and prepare the tailings for subsequent cyaniding. Elsewhere the substitution of copper screens for iron has been practised in the presence of acid water, but that prevents amalgamation in the box. Lime is preferable.

*Speed of Issue.*—Diverse views are held by millmen as to the merits of punched-sheet and wire-mesh screens respectively in relation to the facilities which they afford for issue of pulp. It cannot be denied that in proportion of actual aperture to total area, the former is much inferior to the latter. From this it appears to have been somewhat hastily assumed that the wire-mesh screen must necessarily give the more rapid discharge. No doubt the assumption is correct so far as sheet screens having round punched holes are concerned. But the conditions offered by the slot-punched screen are totally different. An examination of the diagrams representing these screens (p. 85), and still more an observation of the screens in use, will show that the long slots, particularly in the diagonal pattern, favour a stronger current and a greater rush of suspended particles to the particular lines of exit. Though the total aperture area is greater in the woven wire than in

the punched sheet, the interruptions to exit are much more numerous. So that, even with a clean pulp—a mixture simply of water and ground ore—it is at least doubtful whether in practice the wire mesh has any advantage over the slot-punched sheet. But a clean pulp is a rarity. Hardly an instance occurs where more or less fibrous material, from mine timbers, bags, fuse, and so on, does not interfere with the work of the screen, and this, it will be admitted, is far more detrimental to the wire mesh than to the slot-punched sheet.

Examples of procedure at different mills are quoted by authorities as if to show that certain laws have been arrived at. Thus instances are given where a punched screen is used when a retarded issue is desired, and a wire mesh when the increasing percentage of sulphides makes a rapid issue advisable. Rickard is emphatic in claiming that wire mesh is superior to slot screens in rapidity of discharge and consequent reduction of sliming. But it does not appear that any extended experiments were made by himself; he rather quotes the opinions of others. These opinions will be found to be contradictory in many respects, which means that too much reliance cannot be placed on them. Even if they could be regarded as absolutely reliable for the particular ore and mill concerned in the trial, too hasty generalisation from them is to be avoided. Prolonged runs at Lucknow with No. 7 slot-screens and 30-mesh brass wire were distinctly in favour of the former, which gave a more rapid discharge, produced less slimes, and cost per ton little more than one-third as much as the brass wire. The great mills of the Homestake Co., Dakota, in which various new ideas have been tried, remain constant to plain (not burred) slot-punched screens, wire cloth having been proved unsuitable, because it chokes and breaks away with the pressure (Rickard), clearly showing that the issue is less free and rapid. On the other hand, the Rand batteries almost universally employ steel-wire mesh. It is thus evident that there is much to be said on each side, and instead of regarding the question as settled one way or another, the best course to pursue with each ore and mill will be to try

several kinds of screen on their own merits. As has been already said, the cost of screens per ton milled is such a small item as not to be worth consideration ; but even a small percentage of extra output is a matter of considerable moment and deserves every attention.

*Clearing Screens.*—In practically all batteries the screens need constant supervision, not only for wear but for choking, with fine chips &c. Every opportunity should be taken to remove wood-chips from milling dirt before it enters the battery—at mine-shoots, sorting-floors, grizzlies, breaker-shoots and feeder-hoppers—but they cannot be entirely eliminated. The slot-screen is comparatively easy to keep free. A spatula such as is used in cleaning up plates, say 15 in. long and 1 in. wide in the blade, rapidly passed up and down the face of the screen (with the flat side against the screen) serves to shave off as it were the little protruding obstructions, and, by temporarily checking the current, changes the positions of the particles, and breaks up the blockade. A strong harsh scrubbing-brush has much the same effect. Either can be applied outside the screen, and without causing any stoppage of machinery, the action being only momentary. Obstructions cannot be so easily or effectively dislodged from round-punched screens, and are still more troublesome in wire mesh. When the space above the screen is occupied only by a curtain, the millman can reach inside with his spatula and scrubbing-brush. But the efficient clearing of wire-mesh can only be accomplished by removing the screen and drying it before applying the brush, so that extra sets must be always available for replacing those under treatment.

*Splashboards.*—It is a common practice in Australian mills to have a splashboard in front of the screens. This is usually made of sheet iron, and is suspended along its upper edge so as to be readily swung back, being of considerable weight. As a general rule, it completely encloses the screen, and most effectually hides it from the view of the millman. Its object is to prevent the issuing pulp from splashing about ; but this is just as well attained by sheet-iron wings at each side of the top apron plate and mortar lip, which can be

unshipped in an instant, and never offer the least impediment to the millman's watchfulness of his screens.

*Depth of Issue.*—The depth of issue or discharge is the distance between the working face of the die and the bottom edge of the screen. It would be more properly called depth of hindrance to discharge or depth of retention, for it is the measure of the extent to which the pulp is retained in the box. The figures vary widely on different fields, as will be seen from the tables relating to screens (pp. 88, 89), according as a rapid issue and only partial amalgamation in the box are desired on the one hand, or as output is sacrificed to more complete pulverisation and amalgamation on the other. There will obviously be in each individual case a particular depth which will give the best conditions for the results aimed at, and, once this is known, an effort should be made to maintain it as far as is practicable. The constant attrition of the die is of course the disturbing element, and in all well-conducted mills this is counterbalanced in some way. It must be confessed, however, that in very many batteries it is quite ignored, and the depth of issue is allowed to range over the whole distance represented by the thickness of the die, commencing at nil when a new die is put in and gradually growing till it is worn out.

There are several simple ways of controlling the variations. One consists in having wooden cleats of various widths for attachment to the bottom edge of the screen frame, the broadest being added when the dies are new, and replaced by narrower ones as the wear goes on. Another plan is to block up the lower part of the screen to some extent, by fixing sheet-iron blinds of varying width within the frame. When inside amalgamating plates are favoured, the most usual contrivance is a set of chuck-blocks graduated in size. Any of these methods is preferable to elevating the worn dies, which necessitates their complete removal and repacking. In Victoria, it is quite a common practice to raise the worn dies by stowing broken quartz under them, and elsewhere steel false dies are used, sometimes of such a length that one suffices, but more often requiring two.

In very rare instances, the depth of issue is so great that the stamps remain always immersed. This diminishes the rate of discharge.

When direct cyaniding follows the stamp battery, regularity of output becomes a matter of considerable importance, and every effort should be used to maintain the issue at one constant figure, otherwise the cyanide plant will be overcrowded at one time so that tailings may have to be run to waste, or kept waiting at others.

**Dies.**—The die, or false bottom as it is sometimes called, occupies the position of an anvil, the shoe being the hammer. It takes the wear which would otherwise fall upon the bottom of the mortar, and is equivalent to a liner for that portion. At one time, in fact, the die was made in one piece to cover the whole area of the bed of the mortar. But this was an obvious waste of metal when round shoes came into fashion, and was abandoned in favour of a set of dies, each die corresponding to a particular shoe; the latter is now practically the universal custom. At the same time, the old-fashioned long die, generally halved for convenience in handling, survives as a supplement to the thickness of the mortar bottom, being intended not for actual wear, but as a means of giving solidity and height when the die proper becomes worn down. It is thus one of the measures adopted for maintaining an even discharge (see p. 97).

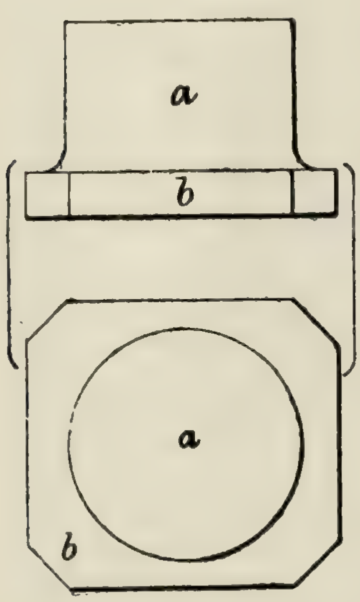



Fig. 35.—Die.

*Forms.*—Dies are made in various forms, the chief distinctions being those which are plain and those which have a base or foot-plate. Fig. 35 shows one of the latter class, consisting of an upper circular boss or die proper *a*, on a square flat foot-plate *b*. The plain die is simply a cylindrical or octagonal boss without any foot-plate. In practice this has several disadvantages, chiefly in connection with proper packing. The bases of the dies are necessarily some distance apart, equivalent

to the free space which must be allowed between the stamp heads to avoid interference, and thus one die does not in the least prevent another from shifting; moreover it is a much more difficult matter to align a number of circles or octagons than a number of four-sided objects, and there is less hold for the packing of ore which is to keep them in place. Especially when new and at their deepest, there is not the same stability; and the incidence of an extra hard or large piece of stone (or drill-head, as sometimes happens) being lodged near the margin of the die, and receiving the full blow of the stamp, will occasionally result in the die being turned completely on edge. These drawbacks are all avoided in the base die. The sum of the bases is computed to very nearly equal the area of the mortar bed, so that each die helps to keep its neighbours in position. Considerable variety is introduced into the shape of the base: thus it may be a simple true square, or the corners may be cut off as in Fig. 35, or so much may be removed as to transform the square into an octagon.

In some batteries, recesses are made in the bed of the mortar to receive the die, which in that case is usually a plain cylinder. Another fanciful form is a cylindrical die with four small feet; and yet another is a bevelled edge to the square base. There does not seem to be any advantage in these vagaries.

A point which is well worth attending to but is nearly always overlooked, is to provide a space for insertion of a chisel bar, by making a bevel in the centre of one edge of the base; this will be found to greatly facilitate removal of the die for clean-ups and when worn out.

The foot-plate with amputated corners gives abundant room for collection of gold or amalgam in the box out of reach of the stamps, and does away with the need for making such accommodation by packing several inches of coarse quartz under the dies, which is not to be commended in any event.

*Material.*—Iron in one form or another is the metal universally employed in dies, but the variety ranges through cast iron, wrought iron, fagoted iron, chilled cast iron, forged steel,

cast steel, chrome steel, and manganese steel. As a rule but little judgment is displayed in selection of metal. Because a hard metal is desirable in the shoe, it is supposed that the same argument applies to the die, whereas its task is quite different. The case is on a parallel with the work done by hammer and anvil, and the latter is always made softer than the former. Similarly the metal of shoe and die should never be alike, but the die always of less resisting material. The die is protected at all times by a layer of crushed ore (that is when automatic feeders are used and kept in proper order), and its life much exceeds that of the shoe. There is room for considerable choice of metal so long as the cardinal point is kept in view. Questions of cost and facility of supply are in many instances the controlling factors in making selections. Thus in California, cast steel dies are placed under chrome steel shoes ; in Victoria, wrought iron under cast iron; at Lucknow, N.S.W., cast iron and forged steel under chrome steel ; at the Homestake Mills, South Dakota, soft cast iron under hard cast iron ; on the Thames, N.Z., unchilled white hematite cast iron under chilled ditto. On the other hand, there are many batteries all over the world, even in the progressive Rand, and at the Alaska Treadwell, where identical metal is used in shoes and dies. At one time, forged steel could not be got of a quality worth putting in, but the forged steel dies now turned out by the Sandycroft Foundry, Hadfields, Firths, Fraser & Chalmers, and other firms, give excellent results, and in the author's opinion it is by far the most desirable material for the purpose ; vibration and shock are notably reduced by its use, and this means a corresponding diminution in breakages, especially of stamp stems, with the attendant delays.

*Dimensions and Weight.*—The diameter of face of the die is adjusted to that of the shoe, and usually equals it exactly, but occasionally it is made slightly wider. This would seem to be liable to cause "cupping" in the die, by which is meant a hollowing out of the face, which good millmen do all they can to prevent, as it reduces its efficiency, and is one cause of broken stems. When a die is badly cupped, it is better to

sacrifice it than to continue using it. At Lucknow, having a lathe equipment, cupped dies are always taken out to have the inequalities turned off, and in this respect forged steel offers another advantage. The diameters of dies range ordinarily between 8½ and 10 in. Their depth or height suffers more variation, the lowest being about 3½ in. and the highest 7 in., while 8 or 9 in. has been suggested. The advantage of extra depth lies in the fact that the discarded or wasted portion, the unworn base, is a pretty constant figure, so that its proportion is lessened as the total depth is increased. The only objection is that a greater difference in the depth of issue is made between new and old dies; but this can be and should be provided for under all circumstances, so that the objection is invalid. Base dimensions run from 9 to 12 in. sq., and from 1 to 2 in. thick, some of the largest dies having the thinnest base. Weight partly depends on material as well as dimensions; about 100 lb. is a common figure, but it reaches from 50 to 150 lb. It should bear a direct proportion to the weight of the stamps, not less than 10 %, unless a false bottom is used under the dies.

*Duty, Wear, and Cost.*—The duty, wear, and cost price of the die are the figures from which its cost per ton milled are to be calculated. Duty and wear are not more affected by hardness of ore crushed than by regularity of feeding. When hand-breaking and feeding are adopted, dies suffer undue wear and tear, in common with every other part of the battery. The amount of metal abraded per ton of ore milled fluctuates between a little under $\frac{1}{5}$ lb. to over $\frac{1}{2}$ lb. The price of metal runs from about 1*d.* to 1¼*d.* per lb. for home-made iron castings, to 5½*d.* for imported chrome steel. The cost per ton milled is mostly in the neighbourhood of ½*d.* to 1*d.*; Clunes wrought iron, with very friable stone, is as low as ⅓*d.*, while in Arizona, with imported chrome steel, it goes as high as 2¾*d.*

Some interesting trials were made by the author at Lucknow, N.S.W., as to the relative cost of iron dies cast at the mine foundry and imported chrome steel dies. The tests extended through the milling of 10,393 t. Local cast-iron

dies weighed new 88 lb. and old 47 lb., it being necessary to remove them before they were nearly worn out, because the mortar beds were over-thin in places; their life was 14 weeks, duty 139·906 t., and wear 41 lb. (46·59 %) or ·293 lb. per ton. The cost new was taken at 15*s.* per cwt., or about 1·66*d.* per lb., and reckoned on the full weight (88 lb.) this equals ·628 lb. or 1·042*d.* per ton; from this is to be deducted the value of the old iron returned, or 47 lb. (53·41 %), equal to ·335 lb. per ton, which was valued at 3*s.* per cwt. or ·321*d.* per lb., and totalling ·107*d.* per ton, so that the nett cost was ·935*d.* per ton milled. The chrome steel dies weighed new 96 lb. and old 40 lb.; their life was 28 weeks, duty 279·812 t., and wear 56 lb. (58·34 %) or ·2 lb. per ton. The cost new was 3·46*d.* per lb., which, reckoned on the full weight (96 lb.), equals ·343 lb. or 1·186*d.* per ton milled, the old metal being disregarded. The steel dies were worn to a thinner condition than the cast-iron, but not down to the base. Chrome steel shoes were used throughout. It will be seen that the figures are slightly in favour of cast iron, in addition to which the work of the mill was more even and satisfactory.

Subsequent trials of forged steel dies established their undoubted superiority over all others.

In 1899 some tests were carried out by G. H. Blakemore, manager of the Prince of Wales mine, New South Wales, on dies of various metals, and by the courtesy of the London Secretary of the Company, Mr. Bethel, the results are tabulated below:

| Material. | Cost. | | Cost per ton milled. |
|---|---|---|---|
| | *s.* | *d.* | *d.* |
| Annealed cast steel (local) . . . . | 23 | 7 | ·292 |
| White cast iron (do.) . . . . | 19 | 9 | ·161 |
| Forged steel (do.) . . . . | 18 | 11 | ·144 |
| Forged steel (Firth's) . . . . . . | 23 | 8 | ·138 |

The durability of steel dies is regarded with disfavour by some millmen, on the ground that amalgam remains in the box unrealised so much longer; and with favour by others, because less labour is entailed in renewing them. There is no justification for either view, if the clean-up is at regular intervals, as it should be; every die is then removed, whether

worn or not, and repacked into position with crushed ore, after all amalgam has been collected from the box.

Below some interesting figures relating to dies are given in tabulated form.

DIES.

| — | Kind. | Diam. | Depth. | Weight. New. | Weight. Old. | Wear per ton. | Duty. | Price per lb. | Cost per ton. |
|---|---|---|---|---|---|---|---|---|---|
| Arizona: | | in. | in. | lb. | lb. | lb. | t. | *d.* | *d.* |
| Mammoth . . | Chrome steel | .. | .. | 120 | 37 | ·35 | 240 | 5·50 | 2·75 |
| California: | | | | | | | | | |
| Angels . . . | Cast iron | .. | .. | 95 | 35 | ·22 | 275 | 2·25 | 0 68 |
| Grass valley . | Do. | 9 | 5 | 93 | 45 | ·49 | 97 | 2·25 | 2·13 |
| North Star . . | Do. | .. | .. | 107 | 40 | ·53 | 127 | 2·00 | 1·68 |
| Colorado: | | | | | | | | | |
| Gilpin County . | Chrome steel | 8½ | 3½ | 55 | 25 | ·19 | 159 | 4·00 | 1·38 |
| Do. . | Do. | 8½ | 3½ | 48 | 26 | ·28 | 78 | 2·00 | 1·06 |
| Dakota: | | | | | | | | | |
| Homestake . . | Cast iron | 8½ | 5½ | .. | .. | .. | .. | 1·00 | .. |
| New South Wales: | | | | | | | | | |
| Lucknow . . | Do. | .. | .. | 88 | 47 | ·29 | 140 | 1·62 | 0·93 |
| Do. . . | Chrome steel | .. | .. | 96 | 40 | ·20 | 280 | 3·46 | 1·18 |
| New Zealand: | | | | | | | | | |
| Thames . . . | Cast iron | 10 | 4 | 108 | 42 | ·47 | 157 | 1·50 | 0·70 |
| Transvaal . . . | Compressed cast steel | 9 | 7 | .. | .. | ·25 to ·33 | .. | .. | .. |
| Do. . . . | Manganese steel | 9 | 7 | .. | .. | ·25 to ·33 | .. | .. | .. |
| Victoria: | | | | | | | | | |
| Bendigo . . | Wrought iron | 10 | 4 | 96 | 26 | ·19 | 375 | 1·28 | 0·31 |
| Clunes . . . | Do. | 10 | 6 | 140 | 30 | ·23 | 470 | 1·23 | 0·36 |
| Ovens . . . | Fagoted iron | .. | .. | 84 | 37 | ·19 | 224 | 1·71 | 0·65 |

**Stems.**—The stamp stem or lifter forms the connection between the shoe which does the work and the tappet by which it is lifted, besides controlling, through the medium of the guides, the space within which the stamp shall fall. It must therefore have sufficient length to reach well above the upper guides, and sufficient girth to give it rigidity.

Additional length is generally provided to enable the stem to be inverted when a break occurs, so that 14 to 15 ft. is a common figure, though it varies from about 8 to 16 ft. Each end is tapered for some 5 in. in length, the diameter being reduced about $\frac{1}{4}$ in. at the extremity, to facilitate fixing in the boss or head.

In most batteries of Australian make, the stem has a coarse thread cut on the upper half of its length, for reception of the tappet, which has a corresponding thread. In this case, the stem is not reversible, and less length is required. But the practice has nothing to commend it, and possesses many drawbacks, so that it is likely to die out; it is unknown outside Australia.

The diameter must be adapted to the strain. With small light stamps, it is often only 2 to $2\frac{1}{4}$ in. In the Transvaal, it has grown from 3 in. when 850 lb. stamps were in vogue to $3\frac{1}{2}$ in. since 1000 to 1250 lb. stamps have come into favour, and may with advantage be further increased.

Delays caused by broken stems are of common occurrence, and materially reduce the efficiency of the battery. Thus in two of the Homestake mills, South Dakota, each of 160 stamps, called the Deadwood Terra and the Golden Star, no less than 20 and 40 stems respectively are broken every month. At the El Callao mill, Venezuela, with 60 stamps, the average number of breaks is nearly 70 per annum, the mean life of a stem being 46 weeks. The Alaska Treadwell 240-stamp mill replaced 125 broken stems in 1899, which is equivalent to an average life of less than 6 months. These breakages have been attributed to crystallisation of the metal due to repeated percussion; but there is absolutely no proof of the assertion, and it seems much more rational to blame it to insufficient strength for resisting unusual strains. The fracture always takes place just where the stem leaves the boss, that is to say where it most acutely feels the lateral or cross strain caused by the shoe striking the cupped edge of the die, or an exceptionally hard or large piece of stone or iron, especially if it be lying near the edge of the die. Stems showing neither flaw nor blemish will sometimes break before

they have done 24 hours' work, and in many instances long ere the supposed crystallisation could have taken place. Stems have a long life in Transvaal mills, because the ore is well broken, regularly fed, and most carefully picked: the shoes and dies wear evenly, and steel drill bits are eliminated. Obviously these conditions, coupled with a sufficiency of metal, are the way to secure a limitation of broken stems.

A good deal of trouble in shifting tappets, and so on, will be saved by having the whole stem turned to gauge, though often a little reduction in first cost is sought by having them cold-rolled simply.

The most suitable metal is a mild hammered steel.

Stems which have had each end broken can be renewed by shutting on a fresh piece, but not many mine smiths can make a sound job of a 3½ in. stem. The weld should be made in one heat, and a steam-hammer is almost a necessity; at Lucknow a most effective substitute for one was made out of a machine drill, run by compressed air.

Some details of stems are given in the table below.

STEMS.

| — | Length. | Diam. | Weight |
|---|---|---|---|
| California: | in. | in. | lb. |
| Brunswick . . . . . . . . . | 15 | $3\frac{1}{8}$ | 375 |
| Douglas . . . . . . . . . | $12\frac{1}{2}$ | $2\frac{7}{8}$ | 290 |
| Electric . . . . . . . . . | $11\frac{2}{3}$ | 3 | 258 |
| Eureka . . . . . . . . . | 14 | $3\frac{1}{2}$ | 450 |
| Lincoln . . . . . . . . . | 13 | $3\frac{1}{8}$ | 320 |
| North Star . . . . . . . . . | 14 | $3\frac{1}{8}$ | 362 |
| Virginia . . . . . . . . . | 13 | $3\frac{1}{8}$ | 320 |
| Colombia: | | | |
| Remedios . . . . . . . . . | .. | .. | 150 |
| Transvaal . . . . . . . . . | 12 to 15 | 3 to $3\frac{1}{2}$ | 350 to 475 |
| Venezuela: | | | |
| El Callao . . . . . . . . . | 14 | $3\frac{7}{16}$ | 392 |
| Victoria: | | | |
| Bendigo . . . . . . . . . | $12\frac{1}{2}$ | $2\frac{3}{4}$ to $3\frac{1}{4}$ | .. |
| Walhalla . . . . . . . . . | $10\frac{3}{4}$ | $3\frac{3}{8}$ | . |

**Bosses.**—The boss, head, or socket, as it is variously called, intervenes between the stem and the shoe, and forms as it were a coupling for them. It is cylindrical in shape, having the same diameter as the shoe, and varies in length from 14 to 23 in., the latter figure being favoured when great total weight of stamp is desired. As shown in Fig. 36, a socket *a* is provided in the upper end for reception of the stem; this is bored to just suit the diameter and taper of the stem, which is kept in place by friction alone, and without any packing between the two metallic surfaces. At the lower end is another

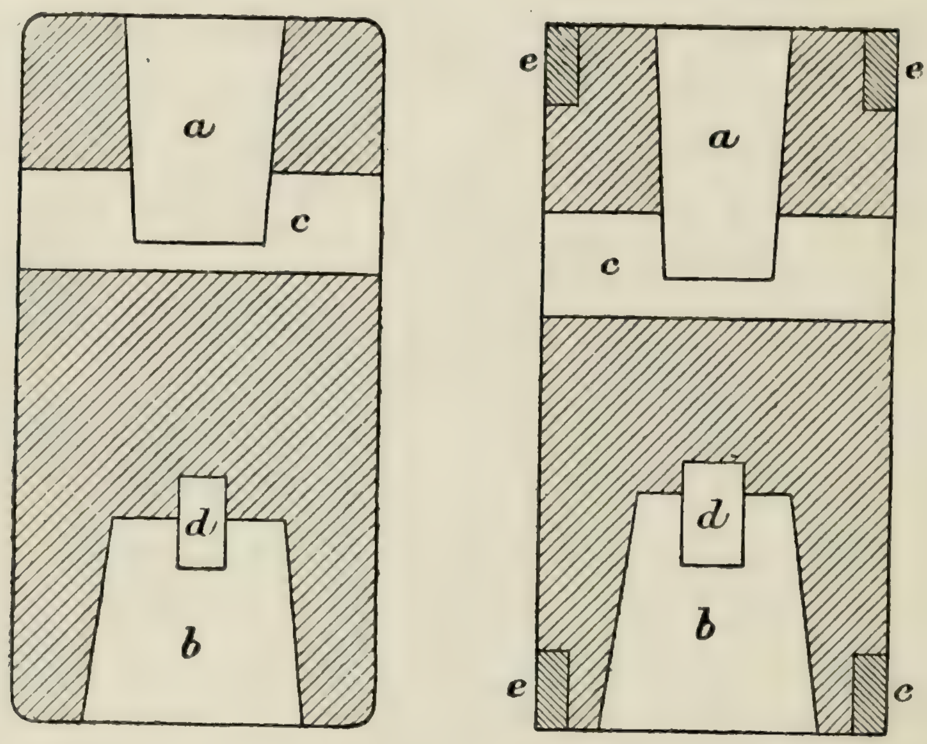



FIG. 36.—BOSS, HEAD, OR SOCKET.

socket or recess *b*, which receives the shank of the shoe; this is purposely made a loose fit and is left rough from the casting, so as to give better grip for the wooden binders by which the shoe is wedged into place. A drift way at *c* permits the disengagement of the stem by driving in a steel wedge; and a similar channel *d* at right angles to *c* serves a like purpose for the shoe shank.

Sometimes the boss is strengthened against the bursting pressure of the shoe shank by shrinking on a 2 by $\frac{3}{4}$ in. wrought iron band, as at *e*, taking care that the boss protrudes a little beyond the band, so that the latter may not be disturbed in case the shoe wears entirely away or breaks off at the shank.

Tough cast iron is a suitable metal for bosses, but, with the tendency to increase weight all round, cast steel is coming into favour. In the Alaska Treadwell new 300-stamp mill, chrome steel is adopted.

With fair usage, bosses should last for years.

Some details of dimensions are given in the table below.

BOSSES.

| — | Height. | Diam. | Weight. |
|---|---|---|---|
| California : | in. | in. | lb. |
| Brunswick | 18 | 8 | .. |
| Douglas | 18 | 8 | .. |
| Electric | 16 | $8\frac{1}{8}$ | .. |
| Lincoln | 18 | $8\frac{1}{2}$ | .. |
| North Star | .. | .. | 220 |
| Virginia | 16 | 8 | .. |
| Colombia : | | | |
| Remedios | .. | .. | 70 |
| Transvaal | 17 to 23 | $8\frac{3}{4}$ to 9 | 260 to 365 |
| Victoria : | | | |
| Bendigo | .. | .. | 159 |
| Walhalla | 14 | 10 | .. |

To fix a boss on the stem, the latter is placed in position between the guides, and the former is stood immediately beneath it on a piece of stout planking lying on the die. The stem, on being dropped, enters the socket *a*, and a few blows administered with a sledge-hammer will drive it in with sufficient force to cause the boss to be raised when the stem is lifted. Care must be taken when using a hammer on the end of the stem, that something is interposed to prevent any injury to the stem, as it has been already "turned" true to fit the boss when the time comes for changing ends. This protection may be a block of wood or piece of thick sheet lead, or a steel cap of exact dimensions. After being lifted and dropped a few times, the stem and boss will be firmly united.

**Shoes.**—Since square shoes have practically ceased to exist even in the backward mills of Australia, there is but one pattern in vogue. This, as shown in Fig. 37, consists of a cylindrical wearing part or butt *a* and a shank *b*. While the pattern is stereotyped, the dimensions vary widely, according as a heavy or light build is desired, though certain proportions are usually observed. Thus the diameter of the butt *a* will be governed by the size of die and of mortar-box, but will not as a rule exceed the depth.

The shank *b* is preferably about 5 in. long, and its diameter at the shoulder should be about half the diameter of the butt, tapering about $\frac{3}{4}$ in. towards the top. This taper must correspond with that of the lower socket in the boss (Fig. 36), and must be carefully proportioned ; if too sudden, it may burst the boss asunder, and will not hold well ; and if too gradual, there will also be difficulty in retaining it in position. The shoulders should curve nicely at *c*, so as to possess as much strength as possible.

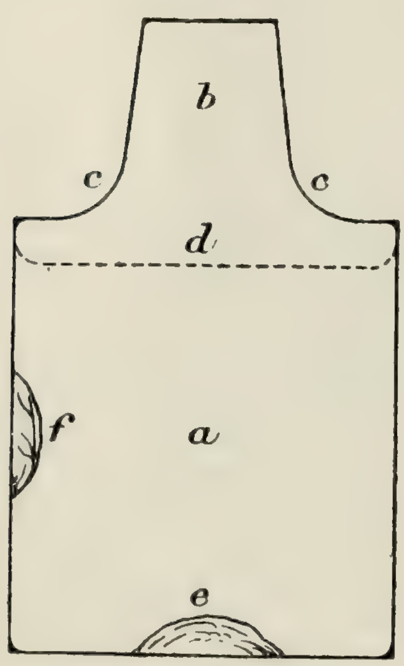


Fig. 37.—Shoe.

At the Utica mill, California, 3 lb. per shoe (1*s*. in value), is saved by casting the shanks hollow—a hole $3\frac{1}{2}$ in. by $2\frac{1}{2}$ in. leaving only a $\frac{1}{2}$ in. shell (W. J. Loring).*

The shoe should never be run until the whole butt is worn away ; indeed, fracture of the thin sole remaining is almost sure to occur before that point can be reached, and it is only good metal that will endure until there is but 1 in. or so left. The dotted line at *d* shows the unworn proportion which is usually sacrificed.

It will thus be seen that a certain economy of labour and time are secured by adding to the depth of the butt, and 12 in. has been adopted in many of the Rand batteries. On the other hand, as wear proceeds, the diminished butt means reduced weight and lessened crushing power or duty, so that

* En. & Min. Jl., Jan. 29, 1898.

provision has to be made for adding weight in some other way, and unless that addition can be made with greater expedition and simplicity than accompany the changing of a worn shoe for a new one, the gain acquired by a deep shoe is not very real.

The compensation of weight for worn shoes has hardly been considered on any field except the Rand, and the methods practised even there are of a makeshift character involving much work. Thus one plan is to have several sizes of boss in reserve, and to change about the shoes and bosses, placing a new shoe on a light boss, and a worn shoe on a heavy boss; this must necessitate a frequent transferring from one to the other, and occupy much time. Another system is the interposition of a false boss, or "chuck shoe" as it is called, between the true boss and the worn shoe; this serves the same end, but requires if anything more time for carrying out. In addition to the time consumed by either method, there is the additional objection to the wear on the sockets by these continual fixings and unfixings, which will ultimately result in bad joints. Neither seems to be as good a plan as that recommended long ago by Prof. Louis, viz. to key iron discs, slotted so as to pass the stem, above the tappets or above the heads, where they can be held on by set-screws. A still simpler arrangement would be to add spare tappets to the stems above the upper guides, provided the stems were sufficiently long.

It would appear that this expedient has not given satisfaction in some Rand mills where it was tried, but J. A. Wilkes* explains the failure by recourse having been had to old tappets of such length that they could not be properly adjusted to the short stems.

The shifting of tappets is a small matter compared with changing bosses. In fact bosses are apt to become very firmly wedged, and to demand extreme efforts to accomplish their removal; small charges of dynamite are occasionally necessary.

Stereotyped habit has fixed the length of shoe at or about

* Jl. Chem. & Met. Soc. S. Africa, Apl. 1898.

a certain figure, and the only tendency towards a change is in the direction of increasing it, with the idea of lessening the frequency of renewal and diminishing the ratio of wasted metal in shank to effective metal in body. It is a question whether this course is a right one. The longer the shoe, the greater the leverage on the shank in case of an uneven blow (on a drill-head or a cupped die), and the more necessity for a heavy and stout shank ; also the greater the duration of the shoe, and therefore the more pronounced the evil effects of lessening weight leading to decreasing efficiency. By putting the permanent weight into the boss or head, much more constant duty would be got from a $2\frac{1}{2}$ or 3 in. shoe, the shank of which could be cast hollow, so that the ratio of waste metal need not exceed that obtained by present methods. This would render possible the adoption of a much more durable metal (viz. manganese steel) in the shoe, so that the changing of shoes would actually not be much more frequent than it is in current practice.

The shoe is attached to the boss in much the same manner as the stem is connected, but as the lower socket in the boss is not bored or made to closely fit the shank of the shoe, a lagging is first wrapped round the latter. This consists of a "bracelet" of little strips of wood about $\frac{3}{4}$ to 1 in. wide and the same distance apart, tied together midway by string, and of such a size that the circle just tightly envelops the shank. The shoe thus prepared is stood in exact position on its die, and the stem and boss are released and let fall upon it. A few blows with a heavy hammer are applied to the top of the stem to make sure (always using a shield between stem and hammer), and then the whole is lifted and dropped a few times. The wooden wedges swell when they become wet, and hold quite tightly. Spare bracelets should always be kept ready for emergencies.

The choice of metal for shoes is as wide as that for dies, and so long as the rule is observed of providing a harder metal for the shoe than for the die (see p. 100) the selection matters but little.

At many mills, ordinary cast-iron shoes are used, and in some cases their adoption is warranted. Where inaccessibility would make imported steel articles very costly, and their supply uncertain, it may happen that home-made cast-iron ones can compete successfully on the score of economy. In such cases, there is often an abundance of local scrap iron to be had for almost nothing, and sometimes the mining establishment will have a foundry of its own. This was the case at the Wentworth mine, Lucknow, N.S.W., and there at times a great many shoes and dies were cast : wreckage from early Colonial machinery was used in the proportion of about $\frac{3}{4}$ to $\frac{1}{4}$ hematite pig. A record of the life of every shoe and die was systematically kept, and from this has been prepared the appended note of the average duration and output (number of tons crushed) in 4 batteries for a considerable period, using home-made cast-iron shoes.

| No. 1. | | No. 2. | | No. 3. | | No. 4. | | Mean. | |
|---|---|---|---|---|---|---|---|---|---|
| days | tons | days | tons | days | tons | days | tons | days | tons |
| 46 | 72·67 | 53 | 109·13 | 40 | 70·81 | 47 | 83·82 | 46 | 86·29 |

The individual variations are exceedingly wide, ranging from 82 days and 144·73 t. to only 20 days and 35·28 t. These differences are due, in the author's opinion, exclusively to the fluctuating quality of the iron, the foundry man being far from expert. At the same time it may be noted that no shoes of local casting ever broke at the shank. They were always poured with the shank downwards, so that it received the best metal, and the grain naturally assumed a longitudinal structure, that is similar to the strains which the metal has to bear. The evil of this method of pouring is that a soft spot occurs at *e*, where the metal has least solidity and most entrapped air. If a lathe is available, it pays to cast the shoe 2 or 3 in. too long, and to "turn" off the surplus.

No doubt the usual method of casting shoes is with the

shank downward, but on one occasion trial was made at Lucknow of some "toughened" cast-iron shoes, made in Sydney, which had evidently been poured at the side, as they presented a scar at *f*. What process had been adopted for toughening, cannot be stated, as it was kept secret by the maker, but it certainly was successful in a measure. Unfortunately, however, in every instance (10 were tried), before the wear had extended sufficiently deep to enable one to judge to what extent the toughening or annealing had penetrated, fracture of the shank occurred immediately above the shoulder. Some were so faulty in this respect that they never did any work at all, breaking at the neck actually while they were in process of being driven into the boss. There seem to have been three reasons for this weakness. Besides a transverse grain in the metal and possibly inferior metal in the shank, due to the lateral method of pouring, probably they were cast in a chill which embraced the whole shoe. This is very objectionable. While chilling of the butt is on quite correct principles, to secure a hard metal, the shank demands toughness rather than hardness, and it should therefore be of soft iron, gradually merging into the hard butt metal. The topmost inch of the butt itself can afford to be softer than the remainder, because it is not expected that the shoe can be worn down below the last inch. Improper chilling presumably accounts for many broken shanks.

The table on p. 113 gives details of a number of shoes in various parts of the world. Local conditions have much to do with choice of material. On the Rand, cast steel, chrome steel, and manganese steel are almost equally favoured, the economical differences between them being of the slightest. The Homestake mills have abandoned chrome steel (Brooklyn) and reverted to their own cast-iron, for reasons unconnected with the respective qualities of the articles. The scrap from chrome steel shoes is on most fields regarded as valueless, foundries being unwilling to buy it. But at Grass Valley, California, this is not the case, and it sells at 1½*d.* per lb. It is of Brooklyn manufacture. The local foundry adds about one-fifth chrome steel scrap to the iron used for making dies,

SHOES.

| — | Material. | Width. | Depth. | Weight. New. | Weight. Old. | Duty. | Wear per ton. | Price per lb. | Cost per ton Milled. |
|---|---|---|---|---|---|---|---|---|---|
| Arizona: | | in. | in. | lb. | lb. | t. | lb. | *d.* | *d.* |
| Mammoth . . | Chrome steel | .. | .. | 132 | 40 | 190 | ·48 | 5·50 | 3·82 |
| California: | | | | | | | | | |
| Angels . . . | Do. | .. | .. | 175 | 40 | 585 | ·23 | 4·50 | 1·35 |
| Brunswick . . | Cast iron | 9 | 10 | 125 | .. | .. | .. | .. | .. |
| Douglas . . . | Do. | 8 | 9 | 115 | .. | .. | .. | .. | .. |
| Electric . . . | Do. | $8\frac{1}{8}$ | 8 | 123 | .. | .. | .. | .. | .. |
| Gover . . . | Do. | .. | .. | .. | .. | .. | .. | 1·75 | 2·00 |
| Grass Valley . | Chrome steel | .. | .. | 152 | 48 | 251 | ·41 | 4·50 | 2·58 |
| Lincoln . . . | Cast iron | $8\frac{1}{2}$ | 7 | 119 | .. | .. | .. | .. | .. |
| North Star . . | Chrome steel | 9 | 8 | 152 | 48 | 229 | ·45 | .. | .. |
| Do. . . | Cast steel | 9 | 8 | 152 | .. | 198 | .. | 3·00 | 2·30 |
| Virginia . . . | Cast iron | 8 | 7 | 110 | .. | .. | .. | .. | .. |
| Colorado: | | | | | | | | | |
| Gilpin County . | Do. | .. | .. | 83 | 27 | 80 | ·70 | 2·00 | 1·91 |
| Do. . | Chrome steel | .. | .. | 111 | 31 | 202 | ·40 | 4·00 | 2·19 |
| Hidden Treasure | Cast iron | 8 | $5\frac{1}{2}$ | 84 | .. | .. | ·70 | .. | .. |
| Dakota: | | | | | | | | | |
| Homestake . . | Chilled cast iron | .. | .. | .. | .. | .. | ·36 | 1·42 | .. |
| New South Wales: | | | | | | | | | |
| Lucknow. . . | Cast iron | .. | .. | 140 | 40 | 80 | 1·20 | 1·66 | 2·73 |
| Do. . . . | Chrome steel | .. | .. | 168 | 34 | 320 | ·42 | 3·46 | 1·81 |
| New Zealand: | | | | | | | | | |
| Thames . . . | Chilled cast iron | 9 | 10 | 170 | 51 | 151 | ·88 | 1·50 | 1·52 |
| Transvaal: | Cast steel | .. | 10 | 180 | .. | 280 | .. | .. | .. |
| | Chrome steel | .. | to | to | .. | to | .. | .. | .. |
| | Manganese steel | 9 | 12 | 230 | 40 | 390 | ·50 | .. | .. |
| Victoria: | | | | | | | | | |
| Bendigo . . . | Cast iron | 10 | 9 | 180 | 38 | 128 | 1·23 | 1·12 | 1·63 |
| Fortuna . . . | Do. | $9\frac{3}{4}$ | 10 | .. | .. | .. | ·92 | 1·28 | .. |
| Harrietville . . | Fagoted iron | $9\frac{1}{2}$ | 9 | 172 | 38 | 207 | ·55 | 1·71 | 1·45 |
| New Chum . . | Cast iron | .. | .. | 180 | 38 | 128 | 1·10 | 1·28 | 1·80 |
| Pearl . . . . | Do. | .. | 9 | 192 | .. | .. | 1·06 | .. | .. |
| Port Philip . . | Do. | 10 | 10 | 196 | 56 | 100 | 1·33 | 1·34 | 2·62 |
| Walhalla. . . | .. | 10 | 9 | .. | .. | .. | .. | .. | .. |

with corresponding durability in the latter whilst avoiding a hardness approaching that of the shoes. At Bendigo, Victoria, and in Gilpin County, Colorado, local cast iron has proved to be cheaper than imported chrome steel, chiefly on account of transportation, customs dues, etc.

At Lucknow N.S.W., comparative tests ranging over a year's milling (10,000 t.) were made between home-made cast iron and foreign (Krupp) chrome steel, with the following results :—

Cast-iron endured 8 weeks, had a duty of 79·946 t., weighed 140 lb. new and 40 lb. old, giving a wear of 96 lb. (68·58 %) or 1·201 lb. per ton milled. The cost new (1·751 lb. per ton) at 1·66*d.* per lb. (about 15*s.* per cwt.) was equal to 2·906*d.* per ton ; less value of old iron returned to foundry, 31·42 % or 44 lb., amounting to ·55 lb. per ton, priced at 3*s.* per cwt. or ·321*d.* per lb. = ·176*d.* per ton, bringing the nett cost down to 2·73*d.* per ton milled.

German chrome steel endured 32 weeks, had a duty of 319·784 t., weighed 168 lb. new and 34 lb. old, giving a wear of 134 lb. (79·77 %) or ·419 lb. per ton milled. The cost new (·525 lb. per ton) at 3·46*d.* per lb. (actual cost delivered at mill), was equal to 1·816*d.* per ton milled, and, the discarded shanks being regarded as worthless for the small local foundry, the nett cost remains at the same figure. Thus the difference in favour of chrome steel was ·914*d.* per ton milled.

Subsequently, extended trials were made of English chrome steel, supplied by Hadfields, Sheffield, in competition with Krupp's, and in every instance the English article proved superior both in duration and in cost, the margin being small in the former particular but quite considerable (over 10 %) in the latter.

For the subjoined results of some very interesting tests made by G. H. Blakemore, manager of the Prince of Wales mine, New South Wales, in Aug.–Nov. 1899, the author is indebted to the courtesy of the London Secretary of the Company, Mr. Bethel. The computations were made on shoes worn till 2 in. of metal remained in the body.

The first test gave the following figures :—

| Material. | Cost at Mine per Shoe. | Average Wear in 4 crushing weeks. | Cost of Metal worn. |
|---|---|---|---|
| | s. d. | in. | s. d. |
| Annealed cast steel (Australian) . | 36 2 | 6·6 | 34 1 |
| Forged tyre steel * (do.) . . . | 56 1 | 3·3 | 26 5 |
| Chrome steel (Hadfield's) . . . | 38 9 | 2·5 | 12 1¼ |

The second test furnished confirmatory results, thus :—

| Material. | Tons milled. | Cost. | Cost per ton. |
|---|---|---|---|
| | | s. d. | d. |
| Annealed cast steel (Aust.) . . | 482 | 25 3 | ·628 |
| Forged steel (do ) . . . . . . | 482 | 19 2 | ·477 |
| Forged steel (Firth's) . . . . | 626 | 15 10 | ·335 |
| Chrome steel (Hadfield's) . . . | 626 | 13 9 | ·263 |

These figures point conclusively to the superiority of Hadfield's chrome steel.

In 1895 and 1896, the author made a long series of trials on Hadfield's manganese steel, with curious results. So far as actual resistance to attrition is concerned, this material proved to be much superior to any other, being at least 30 % more durable even than the same firm's chrome steel. But hardly a shoe was discarded through actual wear. Some split laterally and lost whole slices 1 to 2 in. thick ; others cracked longitudinally ; and from others again wedge-shaped pieces weighing 10 to 20 lb. came away bodily. This peculiar and undesirable feature seems to have developed itself as an outcome of the extreme non-conductivity of the metal towards heat, all sorts of strains and stresses being set up in the cooling of the castings. This was remedied to some extent by the makers introducing a hollow core about 1 in. diam., whereby the thickness of metal to be annealed was reduced ; but still the evil remained in a sufficiently marked degree to

* Two broke at shank and were replaced, though in practice of course they would be regarded as worn out.

amount to a condemnation of manganese steel for this purpose as being unreliable. It has again been suggested, however, to make shoes only 2 or 3 in. thick, supplying the needed weight of the stamp by an extra heavy boss, and in that way to render possible the use of what has proved itself to be by far the most suitable metal for resisting attrition.

At some mills, figures relating to cost of shoes and dies are not reported separately, but are given together. Examples are :—

| — | Metal used per ton milled. | Cost per ton milled. |
|---|---|---|
| India: | lb. | *d.* |
| Mysore Reefs . . . | .. | 3·43 |
| New South Wales: | | |
| Myalls . . . . . | .. | 2·01 |
| Transvaal: | | |
| Geldenhuis . . . | .. | 3·35 |
| Luipaard's Vlei . . | ·72 | 3·10 |
| Robinson . . . . | ·75 | 2·74 |

**Tappets.**—The tappet or collar is an adjustable attachment to the upper part of the stem, by means of which the revolving cam raises the stamp, so that on release it may fall forcibly and repeatedly upon the material to be crushed. Tappets vary in form and in method of attachment to the stem, but the gib tappet as shown in Fig. 38 has become the recognised and approved pattern. It consists of a casting in iron or steel, essentially cylindrical in form, measuring 8 to 12 in. in height and 7 to 9 in. in diam., and centrally hollow, so as to receive the stem. For securing

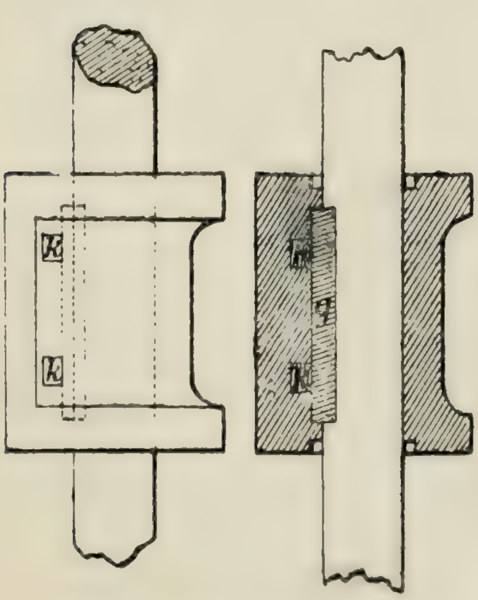


FIG. 38.—TAPPET.

the tappet to the stem, there is a gib *g*, 2 to 2½ in. wide, nearly as long as the tappet, and about 1 in. thick, having its inner

face curved as if forming part of a circle about $\frac{1}{8}$ in. less in diameter than that of the stem, which will give it a more tenacious hold. The gib should be made of forged steel, and its faces and edges should be carefully finished and made true. The top and bottom faces of the tappet are alike, so that it can be inverted when worn, and they are best when the wearing surface slightly exceeds the width of face of the cam working with them. A recess about $\frac{1}{4}$ in. wide and $\frac{1}{2}$ in. deep is turned in the inner periphery of each working face, to facilitate keeping them quite level and uniform. To fix the tappet on the stem, it is dropped into the desired position, the circular channel through it being turned down to just make an easy fit; the gib is then inserted, and the whole is supported and held by hand while keys *k* are driven in by an engineer's hammer, till they force the gib so tightly against the stem that it is firmly secured.

The number of keys used is not uniform, some favouring two and others three, the latter particularly being adopted with the heavier stamps. The keys are of such a length that when driven into position, there is no risk of their interfering with neighbouring tappets; they measure about $1\frac{1}{4}$ in. wide on the broad face, and on the narrow face they taper gradually from 1 in. to $\frac{7}{8}$ or $\frac{3}{4}$ in., the taper being on the side away from the gib. Iron keys seem to have a better grip than steel, but they soon get burred at the small end from the hammering they have to endure; a good plan is to weld on a steel toe.

Cast-iron tappets are quite common, and no doubt, so far as actual wear of the faces is concerned, iron is quite good enough and will last for years, while it is desirable that the metal shall at all events be a little softer than that of the cam, so that the latter may not be sacrificed; but with the big stems needed for heavy stamps and hard ore, it is difficult to get cast iron to resist the bursting strain created by driving in the keys up to real holding point, and for that reason steel tappets are distinctly preferable, though slightly more costly in the first instance. The Alaska Treadwell new 300-stamp mill has chrome steel tappets and cams.

The position of the tappet regulates the drop, or depth of fall of the stamp, and as that should be maintained reasonably constant, a periodical shifting of the tappet so as to effect adjustments is rendered necessary by the wearing down of the shoe. Thus a ready means of fixing and loosening the tappet is most necessary, and the gib and keys have proved eminently simple and satisfactory. Nevertheless, in Australian mills, almost universally another form is adopted, the stem having a coarse male screw cut for a considerable length on its upper part, and the tappet having a female screw to correspond, with keys and lock-nuts to hold it at the desired spot. Not only is the stem incapable of inversion, which in itself is a condemnation of the practice, but the apparent simplicity of the contrivance, and the supposed facility given for adjusting the tappet, are quite negatived by the incessant attention and repairs called for by the vibration of the stamps.

Some figures relating to tappets are given in the table below :—

TAPPETS.

| — | Height. | Diam. | Weight. |
|---|---|---|---|
| | in. | in. | lb. |
| California: | | | |
| Brunswick | 12 | 9 | 125 |
| Douglas | 12 | 8 | 120 |
| Electric | 8½ | 7½ | 83 |
| Eureka | .. | .. | 120 |
| Keystone | .. | .. | 100 |
| Lincoln | 10 | 7¾ | 93 |
| North Star | .. | .. | 112 |
| Stanford | .. | .. | 114 |
| Virginia | 10 | 7½ | 95 |
| Colombia: | | | |
| Remedios | .. | .. | 65 |
| Transvaal | .. | .. | 106 to 130 |
| Victoria: | | | |
| Bendigo | .. | .. | 66 |

**Guides.**—The retention of the stamp in a vertical position during its movement is effected by guides, between which the stem passes. One set of guides is placed below the cam shaft and about 1 ft. above the lid of the mortar; the other set is placed well above the tappet. They are in each case supported by the tie-timbers *d*, Fig. 23. According to very general practice, they are made of wood, and measure about 12 in. deep and 2 in. thick in each cheek. One part of the guide is made in a single piece for the whole battery (5 stamps), and is bolted to the tie-timber; the other part is cut up into as many pieces as there are stems, and these are individually secured to the corresponding parts by bolts (see Fig. 39). In each part is cut a semicircular recess, which forms, when the two parts are put together with the recesses properly adjusted, the hole or stem-way in which the stem travels. The distance apart of the two guides is controlled by wedges inserted between them, and by the holding bolts, care being taken that the stem is free to move and is not pinched in any way. Notwithstanding this precaution, and the abundant application of lubricant, it will be found that there is constant friction between the stem and the guides, owing to the slight thrust caused by the lift of the cam being always on one side, and to the inequalities of the die and its covering of ore.

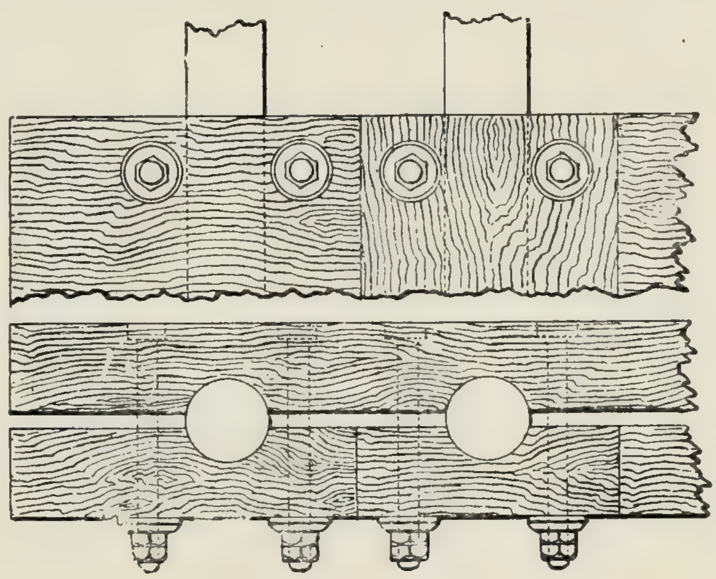
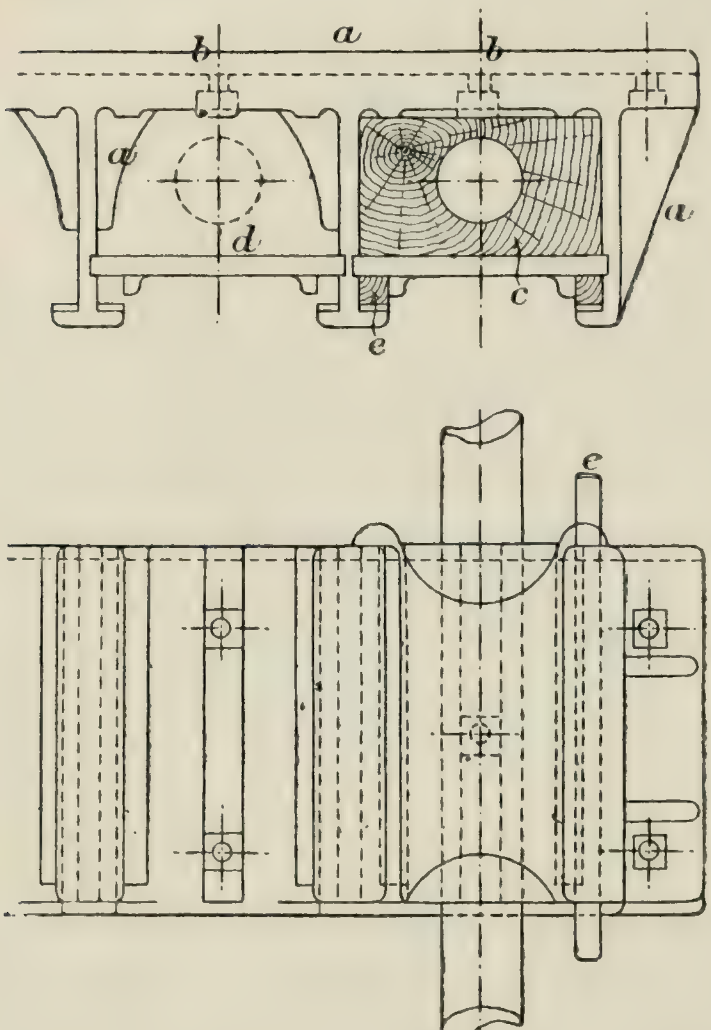


FIG. 39.—GUIDES.

This friction, and consequent wearing away of the guide, is much increased when the passage of the stem is across the grain of the wood instead of with it, and it is therefore distinctly advantageous to cut up each guide into sections, and to place them longitudinally. The guides are then in independent pairs; any stem can be liberated without interference with others, and any guide adjusted or renewed as may become necessary, the removal of two bolts being all that is required.

A further development of this idea is the wooden guide in an iron frame, as shown in Fig. 40. The iron frame *a* is fixed to the tie-timber by a series of through-bolts *b*; the sectional guide blocks *c* are dropped into place, and tightened up as circumstances demand by the follower *d* and long wedges *e*. Vibration makes the wedge keep tight, and a blow from underneath easily loosens it. This is a great improvement on the system of bolting guides in place, for notwithstanding the



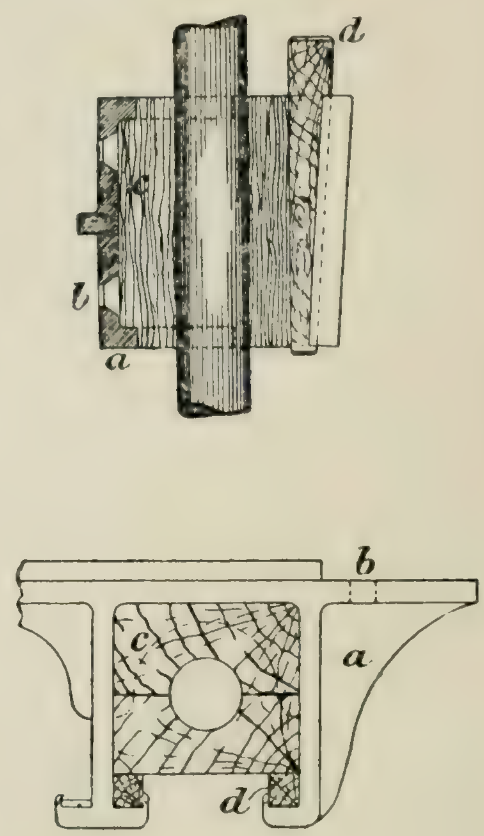




Fig. 40.—Wooden Guide in Iron Frame. Fig. 41.—Ditto.

use of rubber washers under the nuts to take up vibration, loose nuts are of frequent occurrence. Moreover, the guides are in this case simply wood bushings, and their longevity can be much increased by packing behind them, on any side where the wear demands it, thus counteracting also the effect of the thrust of the cam.

Similar ideas are carried out in the guide shown in Fig. 41, where *a* is the cast-iron frame; *b*, bolt-holes for securing it to the tie-timber; *c*, beechwood guide block; *d*, hard oak

tightening wedge. This arrangement is adopted by Bowes Scott & Western.

In selecting a wood from which to make guides, durability seems to be the chief aim. Thus American builders give preference to hard maple, and Australians to red gum and other dense varieties of eucalyptus. Rickard quotes a Victorian mill where red gum guides had lasted for 16 years. At Lucknow, N.S.W., a special timber called tallow-wood is employed: while hard and of beautifully even and straight grain, it has a peculiar greasy feel which gives it its name. It is a native Australian wood, but not common, its price being much above that of other local timbers.

Despite the advantages of wood, the rate of wear is such that renewal of guides constitutes quite an item. In addition, lubrication demands constant attention, and it will be found that, with all precautions, it is nearly impossible to keep the stems really cold; in the author's experience, the stems in most batteries are quite hot when at work, and it is questionable whether this may not be a greater cause of broken stems than the crystallisation which some imagine to take place in the metal. At any rate, it means friction and loss of power.

To find a remedy, quite a number of experiments were made at Lucknow, N.S.W., with iron guides of an improved pattern, though metallic guides are condemned by Prof. Louis as being "apt to work hot and to cut the stems," while their "lubrication is less satisfactory than that of wooden guides." We succeeded in proving the direct contrary, and the cast-iron guide finally adopted, from the design of our chief engineer, Mr. Robert Canning, is so efficient that in two years' work no stem has ever become even warm, no guide has broken or worn in the least, and the consumption of lubricant is reduced by more than half, while its application is simplified. Mr. Canning's cast-iron guide is simply a stuffing box, turned on a lathe so as to be quite true and smooth.

In New Zealand mills * cast-iron guides are a common feature. They are made about 18 in. deep, with a lubricant recess at the top. For dry-crushing stamps, they are simply

* A. H. Bromly, En. & Min. Jl., Nov. 12, 1898.

bored out; for wet mills, they are often run in with white metal. The wear on the stems is not appreciable.

**Cams.**—The raising of the stamp is effected by a cam or wiper operating on the tappet, the object being to convert the rotary motion of the cam-shaft into an upward rectilinear motion of the stamp stem, in such a manner that the lifting velocity shall be uniform. Actual uniformity is, however, impossible of attainment by such means.

When the cam comes in contact with the tappet, the weight is not raised gradually, but is supposed to acquire instantly a velocity corresponding to that of the cam, or, in other words, a blow is struck on the tappet by the cam, which is the cause of much vibration.

The two-armed type of cam is practically universal. In a properly made cam, designed to give a drop of 8 in., the vertical component of the velocity of the cam will be about 1·6 ft. per second for 80 drops a minute, 1·8 ft. for 90 drops, and 2·01 ft. for 100 drops. Having reached a velocity of 2 ft. per second, the stamp will continue unaided to rise a further $\frac{3}{4}$ in., due to excess of momentum over friction. Thus the available limit of speed is governed by the consideration that the upward velocity must not be such as will carry the stamp so high that it cannot reach the die and perform its office before the second arm of the cam comes round to repeat the lift. On reaching its highest point, the stamp commences to fall, its rate of descent being due to the acceleration of gravity, retarded, however, by the friction of the stem between the guides and of the shoe passing through the pulp in the mortar. This retardation amounts apparently to between 7 and 10 %, but will vary materially in proportion to the efficient lubrication of the guides. The maximum practical speed with an 8 in. drop is about 94 falls a minute.

An ordinary double cam is shown in Fig. 42. It consists of a hub *a* and two arms *b*; the latter are faced at *c* and carry a strengthening rib *d*. Cams are often made of cast iron, with a supporting band shrunk on around the hub; but the arduous nature of their work, and the great delay entailed in effecting

repairs and renewals, make cast steel—good open-hearth steel—eminently preferable. The additional cost is more than repaid by increased efficiency and by reduced weight. Not only is the life of the wearing face of the arms prolonged, but less risk is run of fractures, of the arms by blows and of the hub by bursting strains. A cast-iron hub should never be less than twice the diameter of the cam-shaft, and the stiffening rib of the arm must be deep and thick ; while a steel hub demands little more than $1\frac{3}{4}$ times the diameter, and a diminished rib, with no supporting band. It seems surprising

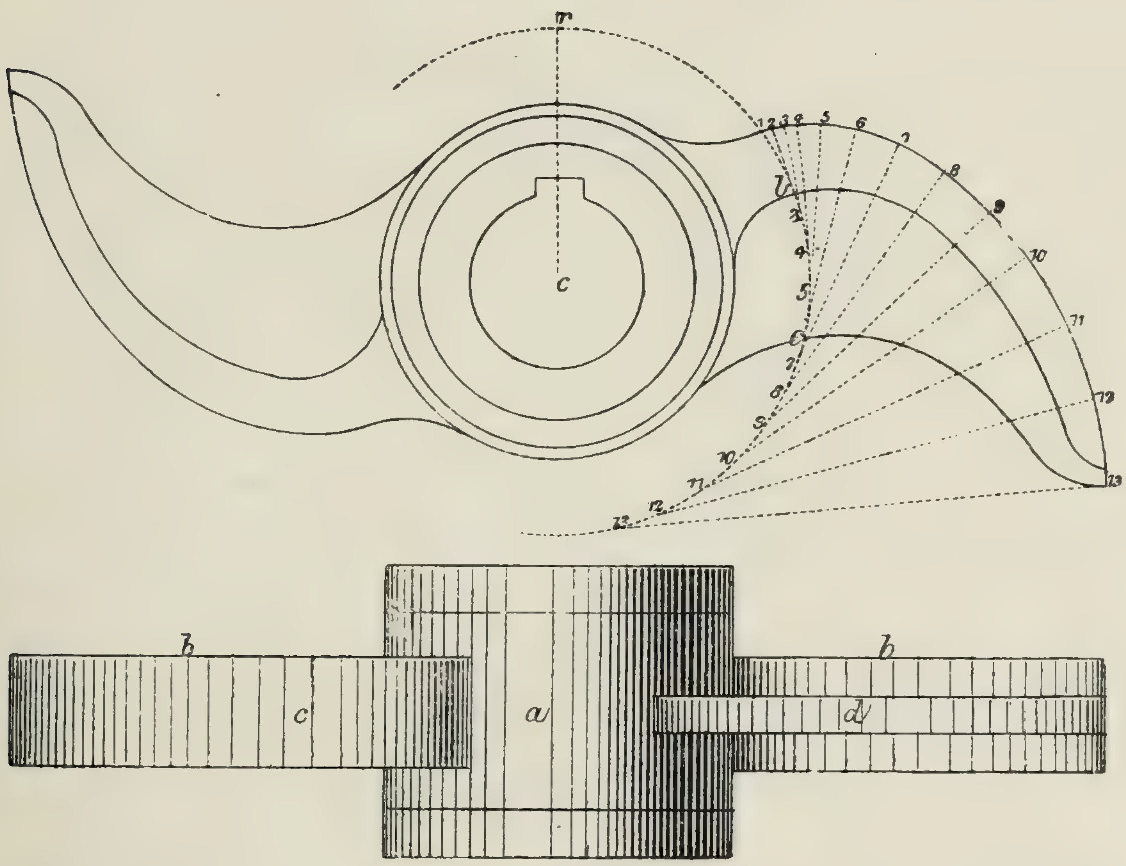


FIG. 42.—DOUBLE CAM OR WIPER.

that a gigantic concern like the Homestake Co., South Dakota, should employ cast-iron cams in their mills, but the question is probably settled in that case by the controlling voice of the owners of the local foundry. In the new 300-stamp mill of the Alaska Treadwell Co., chrome steel cams are used ; in the old 240-stamp battery, 30 cams and 3 cam-shafts were replaced in 1899.

The projection of the hub on both sides of the arms, as seen in Fig. 42, is not now usual ; it is confined to one side, and is generally equivalent to the width of the working face

of the arm, 2 to $2\frac{1}{2}$ in. The side without any projection is that which goes next the stem, and the absence of the projection makes it possible to bring the cam almost into contact with the stem ; it need never be more than $\frac{1}{32}$ in. away from it—just making sure of clearance—and will thus minimise the side thrust. In scores of batteries a full inch intervenes between cam and stem.

*Setting-out.*—The proper curve on the face of the cam, in order that it may perform the required duty with the least friction, is the involute of a circle the radius of which is equal to the distance between the centre of the cam-shaft and the centre of the stamp stem. The cam curve may be constructed on paper by means of tangents, as shown in Fig. 42. If *c* represents the centre of the cam-shaft, and *c r* the radius from the centre of the cam-shaft to the centre of the stamp stem, the circle described about *c*, with *c r* as a radius, is the developing circle of the involute. The distance, representing the height to which the cam is to be lifted, is laid off upon the circumference of this circle, as from the point 1, which distance is subdivided into a convenient number of equal parts, determining, as in Fig. 42, the points 2, 3, 4, to 13. From each one of these points in the circle, a tangent is drawn, on which is laid off a distance equal to the length of arc between the point 1 and the point from which the tangent is drawn. All the points thus determined in the tangent lines are points in the cam curve, and may be connected as shown in the figure, thus producing the line for the face of the cam.

In practice, the line of curvature is produced by cutting from a thin board a circular piece, the radius of which is equal to the horizontal distance from the centre of the cam-shaft to the centre of the stamp stem. At a given point on the periphery of the circular piece is fixed one end of a thread, which must have the length of the greatest desired lift of the stamp, and to the other end of which is attached a pencil-point. The circular piece, with the attached thread wound on the periphery of the circle, is laid on a smooth board, on which the line is to be traced, and the thread, being constantly stretched to its farthest reach, is unwound until it forms a tangent to the

circle at the point where the other end is attached. The line described by the pencil-point is the desired curve. Some makers modify this curve to a slight degree, giving to the cam arm an increased curvature near each of its ends, in order that the cam in its revolutions may come into contact with the tappet at the least practicable distance from the cam-shaft, where the concussion is less than at a greater distance, and to diminish the friction between the extreme end of the cam and the face of the tappet. Such modifications are now quite usual. The extreme end or "toe" of the arm is cut diagonally to a curve equivalent to an arc of the circumference of the working face of the tappet, in order that the whole tappet may clear the cam and fall at once, the clearance taking place the moment the cam toe has passed the centre line of the stamp stem.

*Cam Shaft and Keying.*—The cam shaft is round and of wrought iron (best hammered scrap) or cast steel (open hearth) the latter being far preferable. It is turned true throughout the whole length, and rests in babbited bearings, which are usually supported on shoulders cut in the upright battery posts as in Fig. 25. A cap to the bearing is often dispensed with, when there is no upward strain to resist; but a cover of some kind is desirable for exclusion of dust. Lubrication is a matter of primary moment, as the work of the cam-shaft is very severe, and often the vibration is so great that automatic lubricators cannot be relied on.

Most commonly one shaft serves for two batteries (10 stamps), being supported midway by a bearing on the central post. This arrangement economises space, lessens the outlay on driving pulleys, belting, etc., and reduces the number of bearings to be attended to; on the other hand, in case of a breakdown of cam or cam-shaft, it means 10 stamps idle, instead of 5 when each battery has its own cam-shaft.

As the lift of the cam causes a side-thrust on the stamp, the effect will be transferred to or recoil upon the cam shaft, and must be provided for. This is done by attaching a collar to the shaft just within the bearing which is on the same side of the stems as the cams are, or more simply, when two

batteries are operated by a single cam-shaft, by furnishing one with right-hand and the other with left-hand cams. In the former, the hub is on the right side of the stem when looked at from the feed side of the battery, and in the latter it is on the left side. Thus the thrust of one cam is resisted by the opposing thrust of another. Sometimes on a 5-stamp cam shaft, 3 cams will be right-hand and 2 left-hand.

The keying of the cams upon the shaft is a matter of some intricacy. With light stamps, the cam-shaft will be 4 to 4½ in. diam., according as it serves one battery or two; with medium stamps, it will range between 4½ and 5½ in.; and for heavy stamps, even 6½ in. is not too much. In the Rand mills, 4¾ in. is about the minimum, but 6 in. is very rarely reached, and scarcely ever exceeded, even with 1250-lb. stamps, which seems to give too low a margin. With the smaller cam-shafts, a single key-way suffices, but with the larger shafts and heavier loads two are preferable.

These keyways are cut the whole length of the shaft. But as the stamps are intended to fall in a certain pre-determined order of rotation, and not all simultaneously, each cam will require to have its keyway cut in such a position as to enable it to take its right place in the sequence, and, once this keyway is cut, that cam cannot properly take any other place. The key must be driven in from the hubside of the cam, so that it resists the thrust of the stem.

The great drawback to this manner of securing the cams on the shaft is that the removal and renewal of a damaged cam becomes a very tedious matter, involving the disturbance of every cam between the damaged one and the end of the shaft. Such serious delay is caused in this way that it is often better to let the battery run temporarily one stamp short, until Sunday gives an opportunity for doing the job without hindrance to regular work.

Sometimes the damaged cam is freed by breaking the hub and a sectional cam (in halves) is substituted as a makeshift till the renewal can be properly attended to.

These serious inconveniences attached to the ordinary keying of cams have led to many attempts at improvement, and

complete success has recently been attained by the Blanton patent self-tightening cam, made by Fraser & Chalmers. Whereas ordinary cams have to be sent to the mill in an unprepared state, and much time is then occupied in cutting keyways in precise positions, Blanton cams are delivered in a finished condition, and can be fixed in a few minutes without interference with any other cam on the same shaft. Their reliability and durability are greater also, because the entire cam takes hold upon the shaft, and dependence is not placed on the shearing stress of a key, which is liable to be driven so as to split the hub.

The Blanton cam and the method of applying it to an existing cam-shaft, are seen in Figs. 43 to 46. Fig. 43 shows the cam, eccentric wedge and shaft all in their relative positions as they are placed upon the shaft; *a* being the shaft, *b* the eccentric wedge, *c* the cam, and *d* one of the two pins holding the eccentric wedge in position on the shaft. Figs. 44 and 45 represent the arrangement of cams upon the cam-shaft so as to give the proper drop. Fig. 46 is a jig by which the holes for the pins in the eccentric wedges may be laid out correctly upon the cam-shaft, and drilled. This jig is split on one side, and provided with a bolt *e*, by means of which it can be firmly clamped in position on the shaft; on one edge, lines are drawn, numbered from 1 to 10. Upon one side, two holes are drilled, and fitted with steel bushings *f*, the inside diameter of bushings, and the distance between them, corresponding with the size of the pins in the eccentric wedge, and their distance apart. All Blanton cams are exactly alike—that is to say each cam will take any position upon the cam-shaft, the arrangement as regards drop being made by the location, once for all, of the holes for carrying the eccentric wedges.

In order to locate and drill these holes, proceed as follows: —1st. Use one edge of the existing key seat as a line from which to gauge the position of the jig for doing all the drilling. 2nd. Starting at one end of the cam-shaft (say to locate cam No. 1), slip the jig over the end of the cam-shaft, and bring it to such a position lengthwise of the shaft as will give the proper location for the cam; then revolve the jig upon the

shaft until the point marked "1" on the jig coincides with the edge of key seat. Then simply clamp the jig fast in this position, and drill the two holes to the proper depth through the steel bushings *f*. In regular order from this end of the

FIG. 43.

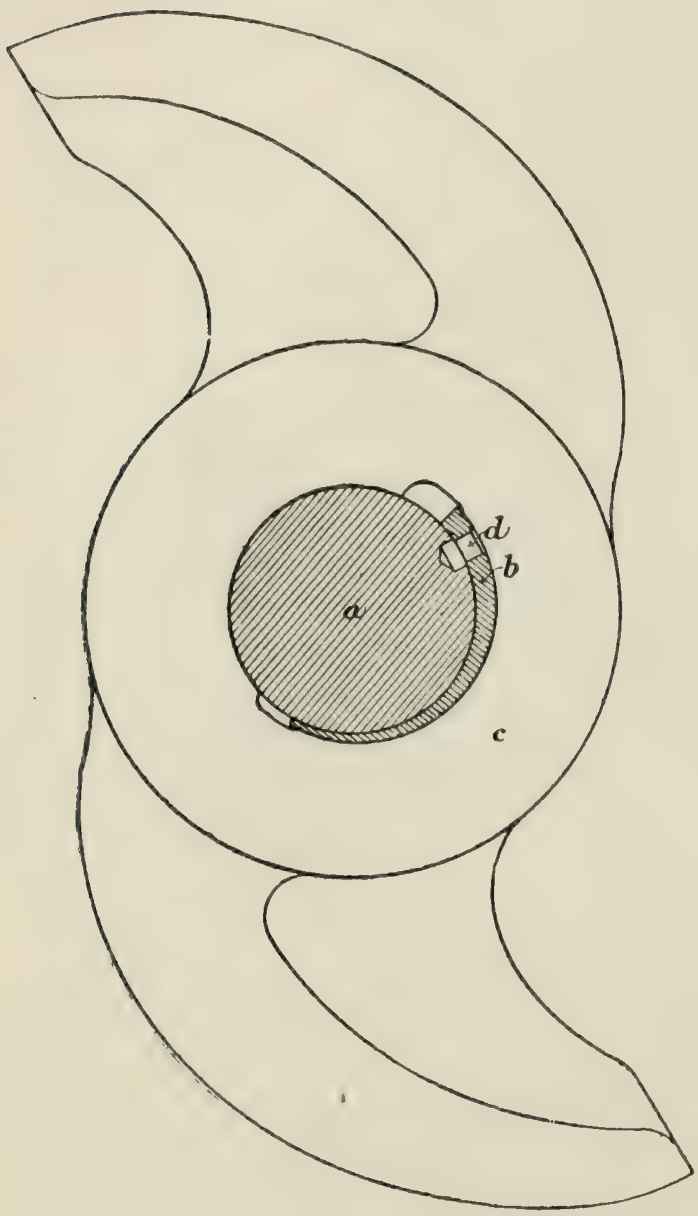



FIG. 44.

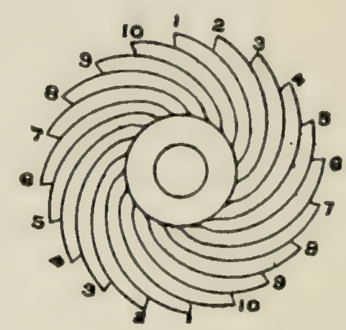


FIG. 46.

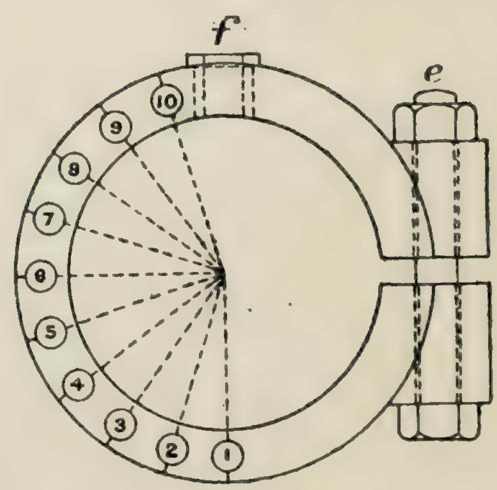
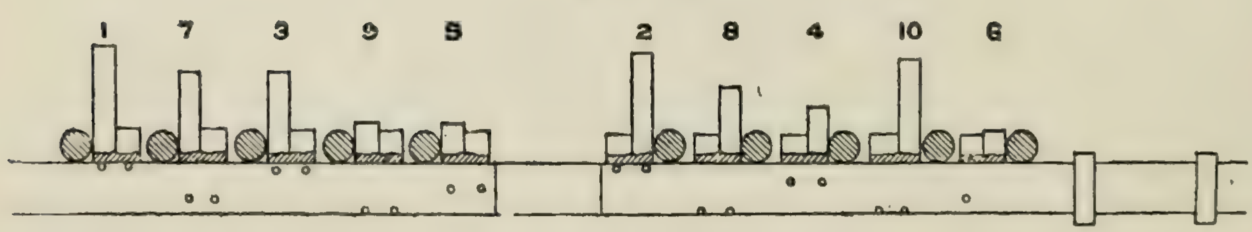



FIG. 45.




BLANTON CAM.

cam-shaft, the next cam to be located would be the one marked No. 7. Proceed for this and all the subsequent cams in exactly the same manner as before described, bringing the required mark on the jig to the same side of the key seat for each

location. In fitting up a cam-shaft for 10 cams, the jig is used in all the positions from 1 to 10, inclusive; while in fitting up a shaft for 5 cams, it is used only in the positions 1, 7, 3, 9 and 5. The numbering here given is that corresponding to the order of drop which Fraser & Chalmers have chosen as a standard. This order is not by any means compulsory, and any of the various sequences described hereafter (p. 145) may of course be adopted.

Of the complete success of the invention, and the advantages it confers, the author can speak from experience, and gladly adds his testimony to that of many others. All mills with any pretence to advancement have the Blanton cam in established use.

When necessary to remove a cam, a slight blow on the back edge suffices to loosen it.

In view of the weakening of the cam-shaft by the holes drilled in it, some increase in its diameter seems advisable.

The same principle may with equal success be applied to fastening pulleys on shafts of all kinds.

In the case of cam-shafts, the idea has been carried further,

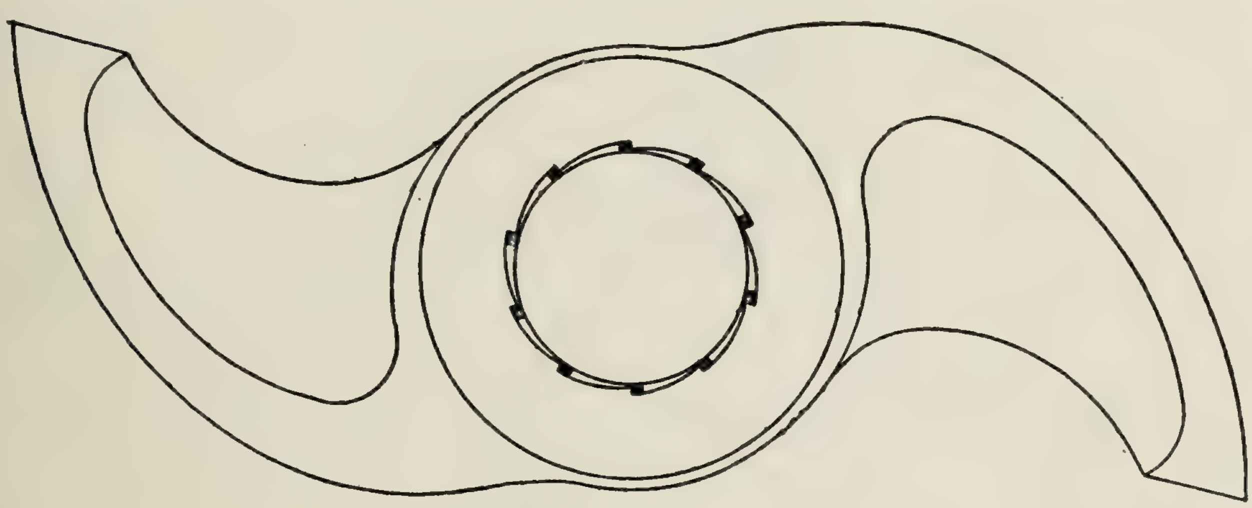


FIG. 47.—BLANTON CAM.

the shaft itself being made of such shape that it obviates the need of any taper bushing, as shown in Fig. 47. Ten taper faces, equidistant, so as to distribute the cams properly for 5 or 10 stamps, as the case may be, are cut on the cam-shaft. The section of the cam-shaft where the cam is secured is the same, whether for 5 or 10 stamps. The taper faces on the shafts and the corresponding faces in the bore of the cams are

made with mathematical precision, by machinery specially designed for the purpose, thus ensuring proper fit and division of the cams on the shaft. Although the cams are located and marked to correspond with Fraser & Chalmers' standard drop, the mill superintendent can arrange the cams for any drop he may prefer, without the slightest alteration, either of cams or shaft. The great advantage of this fastening is the convenience which it provides for the erection and operation of the cams. It is only necessary to place them on the shaft at their respective positions, and tighten on the taper faces sufficiently to hold them in position until put into operation, when they will tighten themselves further on the shaft, in proportion to the amount of work they have to do.

The success of the Blanton cam-fastening has brought a number of imitations into the field.

The Barbour, made by the Risdon Iron Works, San Francisco, is in the form of a curved wedge, on the under side of which is a spline to fit the longitudinal grooves in the cam-shaft. It is used on the principle of the earlier form of Blanton cam-fastener. The positions of the splines on the 5 wedges of a set differ 18° from one another. This, in connection with the two grooves, 90° apart in the shaft, gives all the combinations necessary for the positions of the 5 cams of a battery.

The E. P. Allis Company, Milwaukee, Wisconsin, make a patent "involute" fastener, which differs only in the manner of attaching the curved wedge to the shaft, and does not display any improvement.

In the Davis fastening, also American, the cams are bored out a neat fit for the cam-shaft, and the centre portion is re-bored slightly eccentric. The cam-shaft has a circular milled seat for the wedge-block, the radius of the curve being the same as that of the eccentric portion of the cam. The top and bottom of the wedge-block are finished to the same curve as its seat in the shaft. When the cam is slipped over the shaft and wedge-block, a slight turn back slides the block in its seat until it adjusts itself to a perfect bearing in cam and seat, and wedges the cam securely in place.

All these arrangements appear less simple than the Blanton,

and seem to involve much greater interference with the shafting and more extensive weakening by making recesses for the wedges.

*Mechanical Principles of Cams.*—A comparatively vague idea of the actual work done by the cam existed, until the investigations carried out by Morison and Bremner * cleared up many obscure points, and established definite principles. To designers of batteries, their papers must particularly appeal, but their work is of sufficient interest to those who manage batteries to be worth summarising here.

As already stated, the correct curve for the face of the cam is an involute of a generating circle, the radius of which is equal to the distance between the centres of the cam-shaft and the stamp stem. The properties of this curve are, that the tangent to the curve at the point of its intersection with the stamp axis is at right angles to the latter; that the stamp is therefore, during the axial portion of the lift, raised vertically in the plane of the cam rotation; and that the height through which the stamp is raised, during any given angular rotation of the cam, is equal to the length of the arc of the generating circle embracing that angle. The vertical component of the cam velocity varies directly as the radius of the generating circle and the number of revolutions per minute of the cam, and has the same value at any point of the axial lift.

As the cam face, at the extreme point or toe of the cam, must be of sufficient width to give it the necessary strength, it is obvious that the cam cannot release the tappet until it has swept through a considerable angle beyond the end of the axial lift. The instantaneous vertical component of the linear velocity of the cam toe, whilst passing through the angle corresponding to the angular lift, is equal to the linear velocity of the cam point multiplied into the cosine of the angle included between the cam point and the horizontal centre line of the cam.

In all ordinary cases, the height of the angular lift and the

* D. B. Morison, 'Gravitation Stamp Mills for Quartz Crushing,' Trans. N.E. Coast Inst. Engs., April 30, 1897; D. B. Morison and D. A. Bremner, 'A Development in Gravitation Stamp Mills,' Trans. Inst. Min. & Met., viii. 166 (1900).

velocity due to the involute are such that the former exceeds the momentum lift due to the latter. Therefore the cam toe accelerates the stamp right up to the end of the positive lift, and, in such cases, it is the velocity of the cam toe which determines the actual height of the momentum lift of the stamp.

The lift of the stamp may be divided into three portions,—axial, angular, and momentum,—the two former constituting the total positive lift. During the positive lift, the cam turns through an angle determined by the height of the lift, the radius of the involute, and the distance of the point of release from the stamp axis. During the time occupied by the momentum lift, it turns through a further angle. The stamp then commences to fall, and the angle of cam rotation corresponding to the time of stamp fall depends upon the height of the fall and the angular velocity of the cam. It is evident that if the cam has rotated through 180° by the time the stamp reaches the die, the stamp will be immediately raised again by the opposite arm of the cam, without having time to expend the work accumulated in it upon crushing the ore. Experiments and calculations show that, under the conditions of ordinary stamp-mill practice, the stamp has an interval of rest of about ·1 second after reaching the ore, and this condition would appear to be essential to the performance of effective crushing. Should the cam complete the 180° of rotation, corresponding to one lift and drop of the stamp, before the stamp has completed its descent, it is obvious that the rising cam arm will meet the falling stamp. This action is known to mill men as "camming," and results in breakages of cams and cam-shafts owing to the enormous force of impact due to the added velocities.

A consideration of the above points will make it quite clear that, for a given nominal height of drop, the number of drops per minute, consistent with the avoidance of "camming" and the allowance of the required interval of rest, is limited by natural laws. Other conditions being constant, as the speed rises, the momentum lift, the time occupied by the stamp in falling, and the angle of cam rotation corresponding to the

total lift, increase ; whilst the angle remaining to be swept by the cam during the fall of the stamp, and the time required to rotate through any given angle, decrease. The net result is that the interval of time elapsing between the completion of the stamp descent and the arrival of the cam at the point of pick-up decreases until it becomes a zero or even a negative quantity, which latter, of course, implies that "camming" must take place.

With the object of reducing the momentum lift, and thereby increasing the number of drops per minute practically attainable, it has become customary to modify the involute curve in the vicinity of the toe of the cam, in such a manner as to reduce the radius of the curve at any point as compared with the radius of the involute, and thus to diminish the vertical component of the linear velocity of the point on the curve in contact with the tappet at any given time. This modification does not fulfil the object in view. An unmodified involute cam, of the same extreme radius as the modified cam, will effect a given positive lift with a smaller angle of rotation, whilst the momentum lift in the two cases is practically identical, and the experiments show that a larger number of drops per minute can be obtained with the unmodified than with the modified cam. The only advantage of the modified toe is, that the wear corresponding to the angular lift is distributed over a larger surface ; but as the rate of acceleration of the stamp, and therefore the pressure exerted by the cam on the tappet during this portion of the lift, is small, it is doubtful whether this feature has any practical value. Any modification of the curve, tending to decrease the velocity of lift, necessarily involves an increase in the angle of rotation corresponding to a given positive lift. No advantage is gained, unless the time required to pass through this increased angle is less than the reduction in the time occupied by the momentum lift due to the diminution effected in the final velocity imparted to the stamp by the toe of the cam. In all ordinary cases, this condition cannot be fulfilled.

The number of drops per minute attainable does not increase in inverse proportion to the height of drop, because the

velocities of both the axial and angular lifts vary as the speed of revolution; therefore the momentum lift, which varies as the square of the velocity, becomes greatly increased, and the angle swept by the cam during the momentum lift soon becomes so great that "camming" occurs. This point is clearly illustrated by the investigations.

Incorrect ideas are somewhat prevalent regarding the blow delivered by a stamp. Some writers state that it is a function of the momentum of the mass. The work expressed in foot-pounds expended in raising a body through a given height is, neglecting friction, equal to the weight multiplied by the height in feet, and obviously the force accelerating the falling body is the same as that which had to be overcome in raising it, viz. gravity; therefore the work done by the weight in falling must be equal to the work done in raising it, the distance being constant. When, however, the velocity and weight alone are known, and the height is unknown, it is necessary to express the height in terms of the velocity, and as the height is equal to $\frac{V^2}{64 \cdot 4}$ the formula becomes $\frac{WV^2}{64 \cdot 4}$ = ft.-lb. of accumulated work or kinetic energy.

It is generally assumed that the theoretical maximum number of drops per minute is equal to twice the time in seconds theoretically occupied by a stamp falling *in vacuo* through the given height, divided into 60 seconds; and sometimes the actual number of drops per minute obtained with a cam is divided by the theoretical number of drops thus calculated, and the result is regarded as a figure of merit for that particular design of cam. This would appear to be an incorrect procedure.

The true theoretical basis of reference for cams should surely be a perfect involute cam, with axial release, and without momentum lift, in which case the theoretical maximum number of drops per minute would be obtained by the following formula, where $h''$ is the height of lift in inches, and $r$ is the radius of the generating circle for the involute:—

$$\frac{60 \times 180^\circ - h''\frac{180}{\pi r}}{\cdot 07194 \sqrt{h''} \times 180} = \frac{180 - h''\frac{57 \cdot 3}{r}}{\cdot 21582 \sqrt{h''}} = \text{drops per min.}$$

The vertical component of the cam velocity being the same at all portions of the axial lift, it follows that the cam, when it comes into contact with the tappet at the point of "pick-up," endeavours to instantaneously impart its own velocity to the stamp, with the result that the contact between the cam and the tappet is in the nature of a blow ; as zero time of acceleration involves infinite accelerating force, a time interval has to be created by the yielding of the cam and its supports.

*Rotation of Stamp.*—Not the least important function of the cam and tappet mechanism is the rotary motion thus conveyed to the stamp. This has become a recognised desideratum, and is one of the main advantages possessed by circular shoes over square ones. Thereby the wear of both shoes and dies is rendered much more even, and consequently a better duty is got out of the stamp. The degree of rotation varies with the height of drop (which governs the length of contact between cam and tappet) and the lubrication of the cam and tappet faces. Generally speaking, with 6 to 8-in. drops, a complete revolution of the stamp is accomplished in 5 to 9 drops ; with the deep drop favoured in Colorado, each fall of the stamp is marked by a whole turn. But it is by no means an uncommon sight to see cam and tappet faces so worn and distorted that the rotary motion is reduced to a mere see-saw, or even disappears entirely. The rotation occurs, of course, during the lifting of the stamp, and practically no trace of it is left by the time the stamp shoe meets the ore to be crushed. Consequently the "sliming" or excessive pulverisation of the ore, for which the rotation of the stamp is blamed, cannot be due to this cause. Even with a free-falling stamp in an empty mortar, it will be found that no appreciable trace of a rotary motion can be discerned, and it must be still less when the stamp has had to traverse a retarding medium in the shape of several inches of pulp.

*Side-thrust of Cam.*—It has already been explained that a necessary result of the cam operating always on the one side of the stem must be an uneven lift, producing a side-thrust tending to force the cam away from its work and to make the stamp fall slightly out of the perpendicular. Remedies for

this have been proposed in several ways. One is a cam with forked arms, so that both sides of the tappet are lifted simultaneously, rotation being still preserved by making one branch of the fork slightly longer than the other. A second plan is to use cams having right- and left-hand arms on the same hub, and so placing them that each stamp is alternately lifted on opposite sides ; but this must produce a see-saw and not a rotation. Neither has come into favour.

*Substitutes for the Cam.*—The limitations of speed and weight forced upon the stamp when a cam is used for lifting, as has already been pointed out, have led to various suggestions for replacing the cam by some other form of mechanism. The media most favoured by inventors have been air and steel springs, and quite a number of forms of stamp so actuated have been placed on the market. For various reasons, however, none has been adopted to any extent. Liability to injury, waste of power, limitation of scope, and material modification of the mortar box (interfering with issue and amalgamation), all seem to have been objections combining to hinder the general application of these machines, notwithstanding the admitted drawbacks to the cam action. More recently, Morison and Bremner, whose investigations of the cam and its work have been alluded to (p. 131), applied themselves to the problem, their fundamental object being to provide for a mean velocity of lift exceeding that which is practicable with the cam stamp, in combination with some provision for effecting a gradual acceleration of the stamp at the commencement of the lift, or, in other words, for cushioning the lift. The elements of mechanism finally adopted were a crank and connecting-rod motion, causing a cylinder to reciprocate between guides, the cylinder containing a piston and rod attached to the stamp, and being designed to effect a hydro-pneumatic cushioning of the lift. The axis of the crank-shaft is offset from the axis of the cylinder, with the object of lifting the stamp with a vertical connecting-rod, and causing the angular thrust to take place on the unloaded stroke of the cylinder.

From an engineering point of view, this is a very great step

in advance of previous innovations in stamp battery construction, but industrial considerations may be as fatal to its adoption as to that of earlier novelties. Nevertheless it is worthy of detailed description.

The hydro-pneumatic lifting cylinder, shown in Fig. 48, is of cast-iron, with an annular chamber *a* surrounding the upper portion, and acting as a reservoir for the water used in effecting the cushioning action. The working portion of the cylinder barrel *b* is bored out and fitted with a plug piston *c*, forged solid with the piston rod *d*, which passes through the metallic packed gland *e*, and is connected to the stamp-head by means of a sleeve and stem. The top of the cylinder is fitted with a cover *f*, provided with two lugs to receive the gudgeon-pin and the bottom end of the connecting-rod. Water is supplied to the reservoir *a* by means of a free telescopic pipe connection *h*, communicating with a water channel *i* formed in one of the cast-iron members of the frame of the machine. In order to ensure the maintenance of the requisite supply of water in the reservoir, and to carry off the small amount of heat generated by the cushioning action, the water is continuously circulated by means of a small supply pump ; it is conveyed from the cylinder to a channel formed in one of the lower members of the framework *l*, by means of another free telescopic pipe connection *k*. As there are neither glands nor wearing surfaces in these telescopic pipe connections, it is quite impossible for them to become a source of trouble. In the lower portion of the working barrel is a port *m*, communicating with the reservoir *a*, and in another portion of the barrel, at a slightly lower level than the port, is a passage communicating with the air-vessel *n*. Free communication is provided between the top of the cylinder and the water-jacket *a*, to permit of the transfer of the enclosed air.

The action of the cylinder is as follows :—When the stamp is resting upon the die, and the cylinder is at the bottom of the stroke, the port *m* is open, and the lower portion of the cylinder is filled with water. When the cylinder begins to ascend, the port is gradually closed by the piston, the water is displaced by reason of the relative motion of the cylinder, and

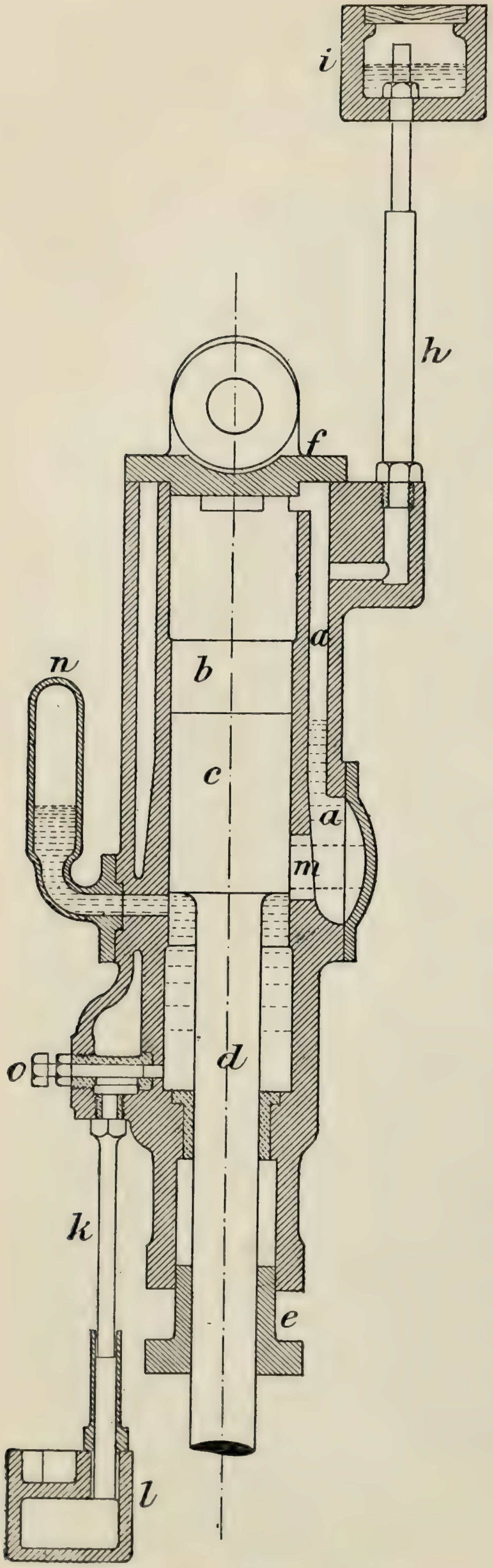



FIG. 48.—MORISON STAMP.

the piston is forced partly into the air-vessel and partly out through the port of gradually decreasing area, at a gradually increasing pressure, until the point is reached at which the pressure below the piston is sufficient to raise the stamp at the then moderate velocity of the cylinder. The cylinder, carrying the stamp with it, continues to ascend at a velocity which increases up to about mid-stroke, and gradually diminishes from that point on to the end of the stroke. The pressure established in the air-vessel, being that due to the maximum accelerating force exerted by the cylinder upon the piston, which occurs at the point of pick-up, it follows that, as the rate of acceleration falls, the air re-expands and does work upon the piston by raising it relatively to the cylinder. When the cylinder reaches the top of the stroke, it momentarily comes to rest, but the stamp, in virtue of its acquired momentum, continues to ascend through a height dependent upon the square of its velocity. Before the stamp reaches the top of its stroke, the cylinder has commenced its descent, and, when the stamp falls, its

motion is not influenced by the motion of the cylinder, because the port *m* has been opened by the before-mentioned relative motions of the piston and cylinder; consequently the water under the piston is perfectly free to escape, as the increasing velocity of the falling stamp causes it to gain upon the cylinder, the velocity of the latter being reduced during the lower portion of its stroke. The stamp comes to rest upon the ore on the die when the cylinder is about $1\frac{1}{2}$ in. from the end of its stroke; it therefore overruns the piston, and opens the port by that amount, thus permitting the water to flow into the cylinder to make up any deficiency: thus the cycle is repeated.

The cylinder, as it slows up, can only retard the stamp through the medium of the friction in the gland and between the piston and the working barrel; but, until a little past the half-down stroke, the cylinder velocity exceeds the stamp velocity, and it therefore, through the medium of the above-mentioned friction, accelerates the descent of the stamp during this period. The net result is that the final velocity of the stamp slightly exceeds that due to gravity, neglecting friction.

By opening the cylinder drain plug *o*, any individual stamp can be quickly, though gradually, brought to rest, whilst the cylinder continues to reciprocate; this without affecting any other stamp. As soon as the plug is screwed up again, the stamp commences to work with a gradually increasing lift until the normal conditions are established.

As the wear of the shoe and die is continually lowering the piston relatively to the cylinder, an easy means of adjustment has been provided, as shown in Fig. 49. The end of the piston-rod *d* is slightly tapered, and fits into a corresponding tapered hole in the top end of the sleeve *p*, being held there by friction in the same manner that a stamp-stem is secured to the boss. As, however, the piston and rod form a comparatively light falling part, a cotterway is provided in the bottom of the piston-rod, so that it can be forcibly drawn into the sleeve, and thus obviate the possibility of any troublesome slackness. In the lower portion of the sleeve is bored a long parallel hole, into which the top end of the stamp-stem *r* fits, and is held in place by a gib and key frictional fastening *s*,

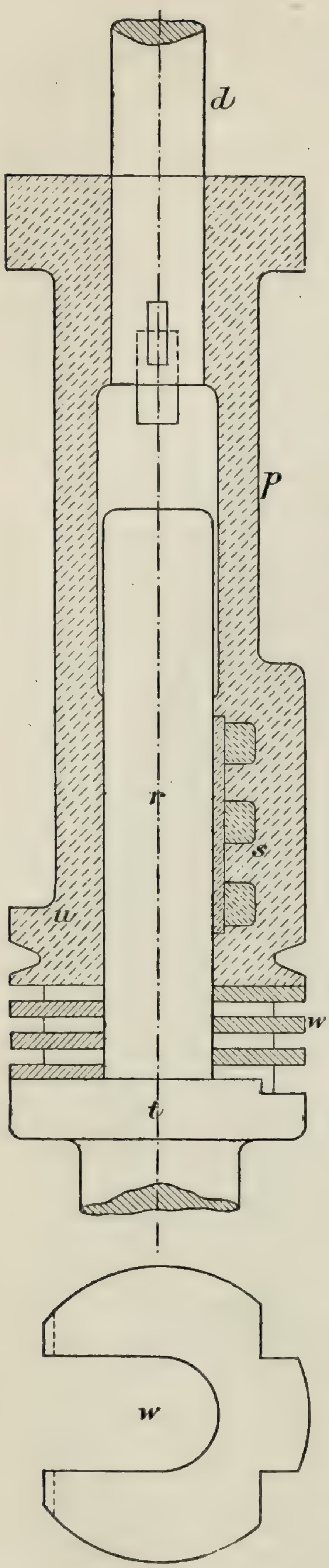



Fig. 49.—Morison Stamp.

exactly similar to that used for cam-stamp tappets. The internal dimensions of the sleeve are such as to permit of the required range of adjustment, whilst ensuring that, when the stem has been withdrawn from the sleeve to the fullest extent necessary, it will still have a bearing over the whole length of the gib. To provide for easy, quick, and correct adjustment for about every ½-in. of wear of shoe and die, and to make quite certain that the adjustment shall not become accidentally altered through slippage occurring between the sleeve and the stem, the latter is forged with a collar *t* upon it at a suitable distance from the end, and between this collar and the bottom flange *u* of the sleeve are inserted, as required, a series of interlocking horseshoe-shape plate washers *w*, each ½ in. thick. All that is necessary is, at the intervals of time corresponding to about ½ in. of wear of shoe and die, to slacken back the 3 keys *s*, raise the sleeve *p*, insert one of the washers *w*, allow the sleeve to drop again, and tighten up the keys. No measurements are necessary.

These washers fulfil the additional important function of compensating for the reduction in the total weight of the stamp caused by the wearing away of the shoe. It was found that, in the case of a 1400-lb. high-speed stamp, the gain in the crushing capacity of a 10-head mill, due to compensation, amounts to no less than 4 t. per day or 5·26 %.

A V-groove is turned in the bottom

flange *u* of the adjusting sleeve, to receive a rope by means of which the stamp is turned. Each set of turning gear (Fig. 50) for one stamp consists of a small rope pulley *a* fitted with a ratchet *b* and carried on a pin *c* at the end of one arm of a bell-crank lever *d*, on the other arm of which is fixed a weight *e* for the purpose of maintaining the required tension on the rope. The bell-crank lever is hung freely upon a shaft common to the set of five turning gears, and to the same shaft is fixed an arm carrying a weighted pawl *f* engaging with the lower portion of the ratchet-wheel *b*. A

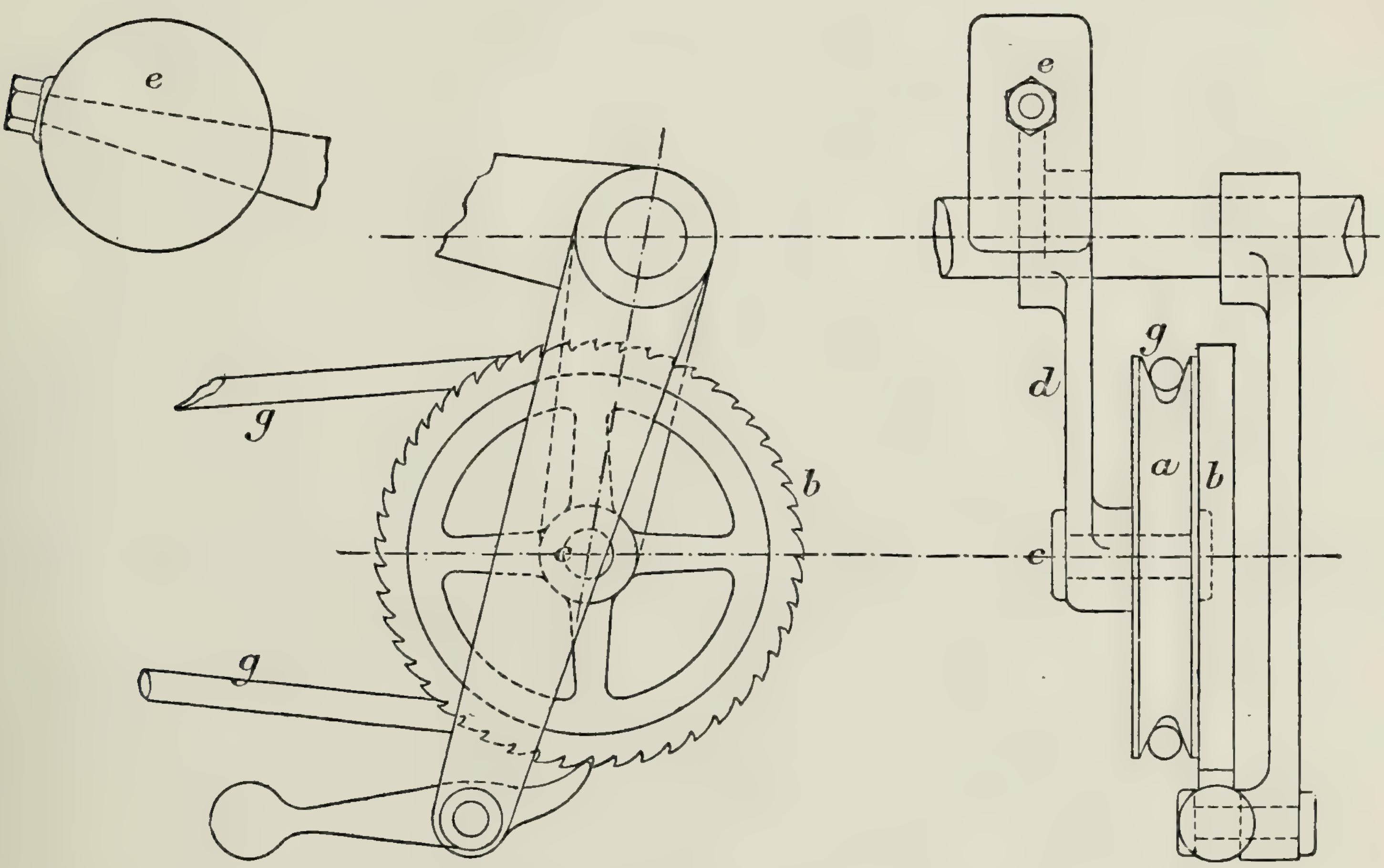



FIG. 50.—TURNING GEAR FOR MORISON STAMP.

piece of ordinary rope *g* establishes connection between the ratchet rope pulley *a* and the rope groove on the adjusting sleeve. By means of this gear, its own reciprocating motion is utilised to effect the turning of the stamp. During the fall of the stamp, the rope pulley is free to swing about the centre of the suspension shaft, and to turn around its own centre, and therefore no transfer of rope takes place to create a turning movement about the axis of the stamp. On the lift, however, the rope pulley is prevented from turning around its own centre by means of the pawl, and consequently a transfer

of rope takes place, to provide for the difference in the lengths of the two sides of the rope, created by the angular motion of the sleeve relatively to the rope pulley ; to permit of this transfer taking place, the stamp has to turn.

In Fig. 51 is shown a typical 10-head cam-stamp mill, for 1150-lb. stamps making 95 drops a minute, converted into a Morison "high-speed" gravitation stamp mill for 1400-lb. stamps running at 132 drops a minute. The mortar, framing, platforms, and feeder remain unaltered, while the old bosses are utilised, the additional weight being partly obtained by means of auxiliary bosses resembling the chuck-shoes mentioned on p. 109. The portions dispensed with are cam-shaft, cams, tappets, upper guides, jack-shaft, and finger-posts. The 5-throw crank *a* is carried in 7 bearings formed in the cast-iron entablature *b*, which is bolted to two special beams *c*, attached to the king-posts of the battery-frame. The cylinders are each controlled by two round guides *d*, connected to the top and bottom cast-iron cross-beams *e*, which are carried by cast-iron side-frames bolted to the entablature. In the case of the relative weights and speeds mentioned above, it is estimated by the inventor that the crushing capacity would be increased as follows :—

$$\frac{1400\text{ lb.} \times 7\tfrac{1}{2}\text{ in. drop} \times 132\text{ drops}}{12} = 115.500\text{ ft.-lb. per min.,}$$

while

$$\frac{1150\text{ lb.} \times 7\tfrac{1}{2}\text{ in. drop} \times 95\text{ drops}}{12} = 68,280 \quad\quad \text{do.}$$

the difference being 47,220 ft.-lb. per minute, or 69 %.

Whilst it will be generally acknowledged that the inventors have shown great ingenuity in contriving a substitute for the cam and tappet arrangement, and that their mechanism places the stamp-battery on a new level from an engineering standpoint, many of their claims for its superiority invite contradiction. Granting the full increase of capacity announced, there is unlikely to be any diminution in first cost of plant for a given output. Buildings will not be reduced in size, because the copper-plate area determines that, and it depends on the

tonnage milled; in fact, an increase is more likely, because inside amalgamation would be impossible in a mortar where such a scour must take place. Power cost will remain the

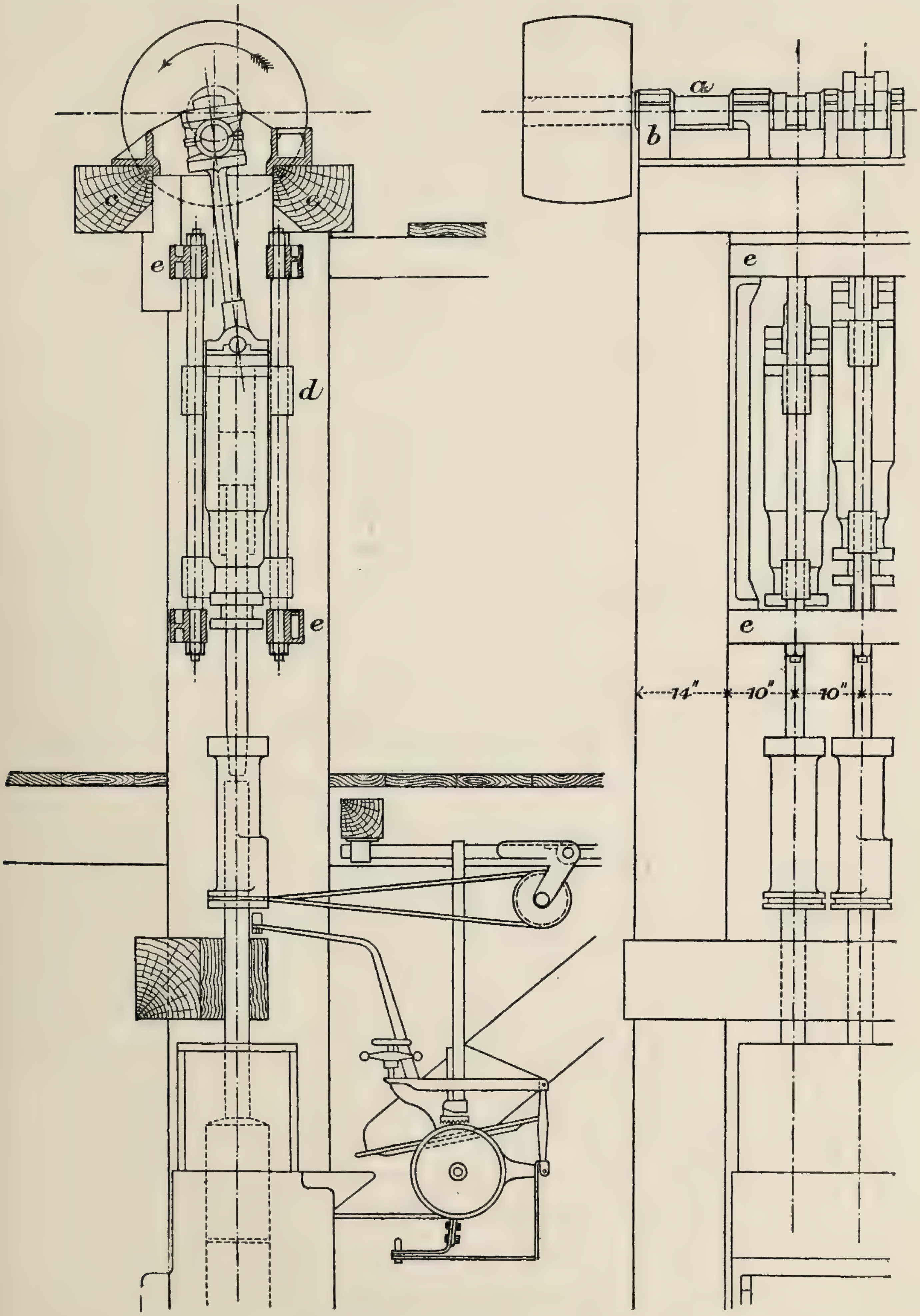



FIG. 51.—ORDINARY STAMP CONVERTED TO MORISON STAMP.

same as before, and labour is likely to be augmented in cost, if not in amount, because there is much more mechanism demanding skilled attention. In fact, the only direction in which an economy may be looked for is in renewals of such wearing parts as cams, cam-shafts and tappets. Throughout the Rand mills, the average cost per ton for maintenance, when it includes wear of shoes and dies, rock-breaker jaws, replacing of stems, cam-belts, etc, is 9*d.* At the Crown Reef, with a duty of 4·795 t., it reaches 8 117*d.* out of a total of 2*s.* 8·13*d.*; at the Robinson mill, "sundry mill spares" (excluding shoes and dies) aggregate 3·962*d.* per ton; at the French Rand mill, all "mill spares" amount to 1·436*d.* per ton. The Mysore mill, India, in 1898, treated 87,155 long tons; out of a total milling cost of 5*s.* 7*d.* per ton, shoes and dies cost 3·8*d.*, and all other "stores" 6*d.* per long ton. At the North Star mill, California, with 850-lb. stamps and a duty of only 1·59 t. through 30 mesh, showing hard rock, the same cams have been in use 9 years, and are still in good order. From these figures it is pretty clear that the cost of renewals in those portions of the battery which are to be replaced by the Morison "high speed" arrangement does not exceed 3*d.* or 4*d.* per ton, so that this figure is the extreme limit of economy which could possibly be effected. But that any such maximum result can be got, is manifestly impossible. There will be wear and tear, with occasional breakages, in the new method as in the old; and instead of the easily renewable cam-shaft, cam, and tappet—rough and crude, but uncostly,—it is proposed to substitute a costly crank-shaft of special design—so special that the depth and order of drop must be determined beforehand, and not departed from,—with numerous wearing parts to keep in order.

**Order of Drop.**—The order in which the stamps shall be made to fall is a matter of considerable importance, and is governed by two main principles—(*a*) that the work of raising the stamps shall be uniformly distributed on the cam-shaft, so that the weight lifted shall be, as nearly as possible, the same at any period of the revolution; and (*b*) that each stamp shall

fall effectively upon the material to be crushed, and maintain a proper distribution of the pulp in the box, and a regular wave from end to end, so as to induce a constant issue without undue splash. Therefore, no two stamps should ever fall simultaneously, neighbouring stamps should not fall in succession, and while any stamp is falling the next stamp should be rising. A proper order of drop is also favourable to uniformity of wear in shoes and dies, by equalising their duty; and to endurance of mortars and foundations.

In Australian practice, these fundamental laws are mostly ignored, and the sequence of fall is usually haphazard. Indeed it is not uncommon to find a deliberate departure from first principles, two stamps being made to drop simultaneously, thus 1 and 5, 2 and 4, 3, that is, commencing with the two outside stamps and concluding with the middle one. No attempt is made to rationally explain the reason for this custom. On many fields the order adopted is 1 5 3 2 4, 1 5 3 4 2, 5 1 3 4 2, or 1 3 2 4 5, all of which are in opposition to the principles enunciated above, and present no redeeming advantage. The favourite sequence in the Rand mills, and the one which has been chosen as a standard by Fraser & Chalmers, after many years' experience, is 1 3 5 2 4, and this has been found to give the most regular pulsation and greatest output. Another order common in the Transvaal is 1 5 3 2 4, which closely resembles the 1 5 2 4 3 of some of the Homestake batteries; the latter particularly causes great wear of screens and chuck-blocks at the point opposite No. 3, because of the almost simultaneous drop of two stamps, but it is credited with securing a maximum duty. Perhaps the most general succession, and best adapted to facilitate amalgamation in the mortar, is 1 4 2 5 3; this is quite usual throughout the United States and South Africa.

**Height of Drop.**—The amount of drop or fall given to the stamps is always calculated on the distance traversed by the bottom of the shoe from the commencement of its fall to the face of the die. Obviously this is only the nominal drop, because a battery is not run empty. Usually the layer of ore on the dies will be 1 to 1½ in. in thickness, and it is on this

that the shoe falls, or should fall, so that the effective drop will be that much less than the nominal drop. With hand feeding and no breaker, the difference is liable to be very much greater at times, and very much less when the feed is allowed to run down.

The nominal drop is fixed by placing wooden blocks of the desired depth upon the dies, and allowing the shoes to rest on them while the tappets are secured in the positions indicated.

It is a common practice to give the two end stamps about ½ to 1 in. more drop than the others, to assist in preventing an accumulation of ore at the ends of the box where, of course, the opportunities for discharge are restricted ; but obviously this possesses the grave drawback that it reduces the speed at which the battery must run, and so diminishes the effective duty.

The nominal drop provided for at various batteries differs most widely. Clearly it must bear a definite relationship to the speed of the stamps, or the frequency with which their fall is repeated, and these figures are controlled by the capabilities of the cam (see p. 132). Moreover the weight of the stamps will exercise an influence, because a greater number of foot-pounds of kinetic energy may be gained by means of weight than by height. The greatest duty—which means not only crushing power, but also discharging power, the latter being quite as important as the former—is gained by maximum speed and weight, the height being made to conform to the speed. Therefore, when inside amalgamation is not specially sought, the tendency is to increase weight and speed and reduce the drop to the minimum compatible, 6 to 9 in. being about the average range. When inside amalgamation is a primary desideratum, the converse holds good, the object being to retain the ore in the box, and give opportunities for a commingling of the gold and mercury, without undue sliming ; in such cases, a drop of 16 to 20 in. is not unusual.

The table on pp. 148–150 shows the figures current at a great number of mills.

**Speed.**—The speed of a battery, or number of drops per minute which can be made by each stamp, is dependent upon

the height to which the stamp is lifted (see above), and both are subject to the limitations of the cam (see p. 132). Practical deductions from experiments and everyday experience are that the maximum number of drops possible with a 7-in. effective or 8½-in. nominal drop, allowing for an "interval of rest" of ·1 second after each drop, is 100, and in actual work it would not be permissible to exceed 96 or 97 drops, as a margin must be allowed for irregularities in the speed of the driving engine and for variation in the thickness of ore upon the dies. With a lessened drop, greater speed is of course possible, and 105 is sometimes reached; while the excessive drop favoured in some Colorado mills brings the speed down to 30.

The "interval of rest" spoken of above is really the time occupied by the rebound of the stamp on striking the ore on the die, and is proportionate to the height of drop.

The table on pp. 148–150 shows a wide range of speed.

**Weight.**—The weight of a stamp is understood to embrace stem, tappet, boss, and shoe when in a new and unworn condition. It is hardly necessary to insist that increase of weight means an augmentation of crushing power, but very little has been done to establish any sort of rule for the rate of progression. The only available figures are those arrived at by Morison and Bremner (see p. 131), from which it would seem that for every 100 lb. additional weight the duty is advanced at the rate of ·4 to ·6 t. per stamp per 24 hours. These figures, however, were obtained with stamps of extreme weight, ranging from 1250 to 1568 lb., and operating on whinstone. Quite different conditions exist in the ordinary battery working on gold-bearing ore, which is far from being a homogeneous rock. In most milling dirt there is an easily friable portion and a hard or tough portion, and the relations between the two are most inconstant, so that it would be very difficult to formulate any rule.

In choice of weight, different fields follow different lines. The high-drop Colorado mills favour about 550 lb.; the Australians range from 600 to 900 lb., without any particular reason; the Homestake and most Californian batteries adopt

850 to 900; while the Transvaal establishments run from 950 to 1250 lb. and upwards. It is rather singular that the Californians have reverted from 1000-lb. stamps back to 850-lb. as a rule, while the Transvaal mill-men are replacing their 950-lb. stamps by 1250-lb. The explanation may be that the Pacific mills make a greater point of amalgamation inside the mortar.

At the Big Cañon mill, California, the weight was raised to 1400 lb. per stamp by adding heavy bosses at the upper end of the stems; but after a short trial they were removed, because, it is said, the "jar caused frequent breakage of cam-shafts."

Success has attended the experiment of making the end stamps heavier than the others with a view of overcoming the tendency to "banking" at the ends of the box. In this way the speed is not interfered with, as it is when the end stamps are given a longer drop.

The appended table of drop, speed, and weight adopted in various mills throughout the world shows how widely the figures vary.

DROP, SPEED, AND WEIGHT OF STAMPS.

| — | Drop. | Speed. | Weight. |
|---|---|---|---|
| Alaska: | in. | drops per min. | lb. |
| Mexican | 7½ to 8½ | 97 | 1020 |
| California: | | | |
| Clinton | 6 | 85 | 1000 |
| Empire | 7 | 90 to 95 | 850 |
| Eureka | 9 | 80 | 950 |
| Gover | 6 to 7½ | 96 | 800 |
| Grass Valley | 10 | 61 | 850 |
| Do. | 10 | 68 | 700 |
| Idaho | 9 | 80 | 950 |
| Kennedy | 6½ | 83 | 850 |
| Keystone | 6 | 75 to 90 | 750 |
| Metacom | 10 | 90 | 900 |
| North Star | 6 to 8 | 82 to 85 | 850 |
| South Spring Hill | 7½ | 93 | 750 |
| Wildman | 7 | 92 | 750 |
| W.Y.O.D. | 6 | 90 | 750 |

DROP, SPEED, AND WEIGHT OF STAMPS—*continued.*

| — | Drop. | Speed. | Weight. |
|---|---|---|---|
| Colombia: | in. | drops per min. | lb. |
| Remedios . . . . . . | 8 | 95 | 350 |
| Colorado: | | | |
| Gregory Bobtail . . . | 16 to 18 | 27 to 30 | 550 |
| Hidden Treasure . . . | 16 to 18 | 30 to 32 | 550 |
| New York . . . . . . | 18 to 20 | 26 | 600 |
| Prize . . . . . . . | 15 to 17 | 28 to 30 | 500 |
| Randolph . . . . . . | 16 to 18 | 30 | 500 |
| Dakota: | | | |
| Homestake . . . . . | 8 to 9½ | 85 to 90 | 850 to 900 |
| Mexico: | | | |
| Mesquital . . . . . . | 8½ | 80 | 650 |
| New South Wales: | | | |
| Lucknow . . . . . . | 6 to 7 | 95 to 98 | 850 |
| New Zealand: | | | |
| Cambria . . . . . . | 9 | 76 | 620 |
| Comer . . . . . . . | 6 | 63 | 840 |
| Kuranui . . . . . . | 5½ | 70 | 670 |
| Moanataiari . . . . . | 8 | 66 | 650 |
| Phœnix . . . . . . | 7½ | 78 | 800 |
| Premier . . . . . . | 7 | 77 | 750 |
| Reliance . . . . . . | 7½ | 75 | 850 |
| Saxon . . . . . . . | 8 to 11 | 72 | 785 |
| Norway . . . . . . . | 8 | 80 | 750 |
| Transvaal . . . . . . . | 7 to 9 | 95 | 950 to 1250 |
| French Rand . . . . . | 8 | 95 | 1000 to 1250 |
| Geldenhuis . . . . . | 8 | 95 | 1250 |
| Robinson . . . . . . | 8 | 95 | 1250 |
| Sheba . . . . . . . | 8 | 95 | 1250 |
| Village . . . . . . . | 8 | 95 | 1250 |
| Utah: | | | |
| Eureka . . . . . . . | 7½ | 100 | 950 |
| Victoria: | | | |
| Ararat . . . . . . . | 7½ to 10 | 60 to 72 | 560 to 756 |
| Ballarat. . . . . . . | 7 to 10 | 50 to 85 | 448 to 952 |
| Beechworth . . . . . | 5 to 14 | 40 to 90 | 476 to 868 |
| Bendigo . . . . . . | 6 to 18 | 25 to 75 | 560 to 896 |

DROP, SPEED, AND WEIGHT OF STAMPS—*continued.*

| — | Drop. | Speed. | Weight. |
|---|---|---|---|
| Victoria—*continued.* | in. | drops per min. | lb. |
| Britannia . . . . . | 8 | 60 | 1050 |
| Castlemaine . . . . | 6 to 15 | 35 to 75 | 504 to 896 |
| Catherine . . . . . | 9 | 69 | 750 to 900 |
| Fortuna . . . . . | 8 to 9 | 68 to 75 | 900 |
| Gippsland . . . . . | 7 to 10 | 60 to 80 | 672 to 840 |
| Harrietville . . . . | 8 | 70 | 700 |
| Hillsborough . . . . | 9½ | 60 | 784 |
| Maryborough . . . . | 6 to 22 | 50 to 75 | 504 to 896 |
| New Chum. . . . . | 9 | 72 | 895 |
| New Normanby . . . | 7 | 60 | 784 |
| North Cornish . . . | 8 | 72 | 784 |
| Oriental . . . . . | 9 | 55 | 784 |
| Pearl . . . . . . . | 7¼ | 74 | 840 |
| Port Phillip . . . . | 8 | 75 to 82 | 728 |
| Do. . . . . . . | 8 | 75 to 82 | 896 |
| Railway . . . . . | 9 | 60 | 720 |
| South St. Mungo . . . | 9⅝ | 68 | 965 |
| Star of the East . . . | 8¼ | 73 | 1008 |
| Stephen . . . . . | 9½ | 50 | 840 |

**Props.**—When it becomes necessary to "hang-up" a stamp, so that its cam may revolve without reaching the tappet, it is supported by a prop, stud, or finger-post, as at *x* Fig. 23. The lower end of the prop is pivoted on a small "jack shaft" fixed across the battery from end to end, and resting in boxes which are fastened by coach-screws to the opposite faces of the battery posts. Each stud is just long enough to support the stamp, when placed under the tappet, at a height of ½ to 1 in. above the maximum reach of the cam. The props themselves are of good sound wood, 3½ to 4 in. sq. at the base, and tapering down to 2 or 2½ in. sq.; they are fixed in cast-iron sockets fitting the jack-shaft, and are often iron-bound at the top, and provided with an iron loop handle. When not in use, they tilt back, as shown in Fig. 23. To bring them into operation, the workman takes one at a time in

his left hand ready to push it forward, while with the right he slips the "fid-stick" between the rising cam-face and the tappet, whereby the tappet is raised to an additional height equivalent to the thickness of the fid-stick, thus giving opportunity for the prop to be pushed underneath it. The fid-stick, which is usually a small piece of quartering, fashioned into a handle at one end, and covered with old leather or vanner belt at the other, should be of the same width as the face of the cam, about $1\frac{1}{2}$ in. thick, and of convenient length (say 15 in.). The operation is simple, but requires "knack" to avoid serious strains and jerks to the arms; the tendency of the cam to pull the fid-stick away from the holder must be resisted with some force, and the prop must be slipped in place with smartness. Using the knee to push forward the prop is a very bad practice, and likely to result in personal injury. When the stamp is to resume work, the fid-stick again comes into play, the prop being quickly withdrawn while the tappet is supported by the fid-stick and cam.

In batteries of Australian build, a more common arrangement is two parallel fixed bars built into the framing, and a third movable cross bar which is laid across them.

**Feeding.**—The regular and steady supply of ore to the battery, commensurate with the rate at which it is crushing and discharging, is a matter of considerable importance, affecting as it does not only the duty or capacity of the battery, but also the efficiency of the amalgamation which is going on inside the box, and still more the life or durability of almost every part of the machine.

Originally all feeding was by shovel, and in the most backward gold-mining camps the practice is still general, apparently for the principal reason that, as no breaker is used in such cases, the man or boy who does the spalling at the feed bin can also wield the shovel. When circumstances absolutely preclude using a breaker—and it may be doubted whether such ever occur—there arises some justification for such a proceeding, but in reality it is adopted from a false notion of economy. Even when the duty per stamp is so low that one

man can feed 5 batteries (a most unusual condition), his wages will amount in a year to very much more than the prime cost of installing an automatic feeder at each battery ; and when, as is usual, one man is needed for every 10 stamps, and that man only receives the lowest labourers' wage, his work is ridiculously costly. And worse than its costliness is its inefficiency. The battery which is not worth furnishing with a

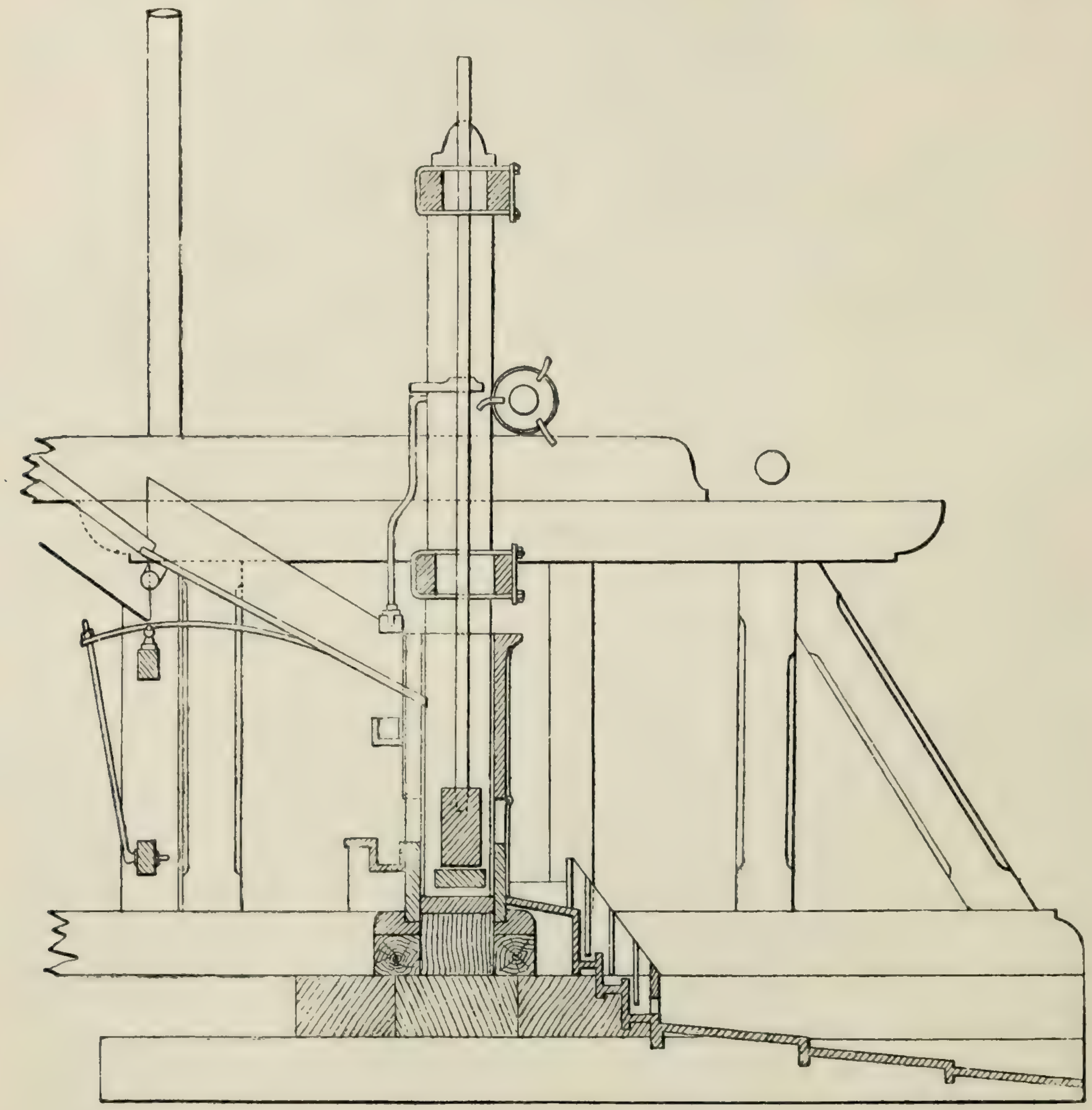


FIG. 52.—SELF-FEEDING MECHANISM.

breaker and automatic feeders is not worth building at all. The few pounds saved in first cost will be paid away over and and over again in wages, fuel, and renewals inside of twelve months.

Hand breaking and feeding constitute the greatest blot on Australian gold milling, yet, strangely enough, one or two

Victorian batteries were among the first to adopt automatic feeders. An example is shown in Fig. 52, as employed at the Port Phillip mill, Clunes, where it has done excellent duty for many years, its cost being less than 10*l.* The shoot leading into the mortar is hinged to the lower edge of the ore-bin, and is set at a less angle, so that it acts as a check to the forward passage of the ore. To the lip of the shoot is attached a vertical rod, of such a length or height that, when the mortar requires replenishing, the tappet on the stem of the central or feed stamp strikes it, thereby jerking forward a certain amount

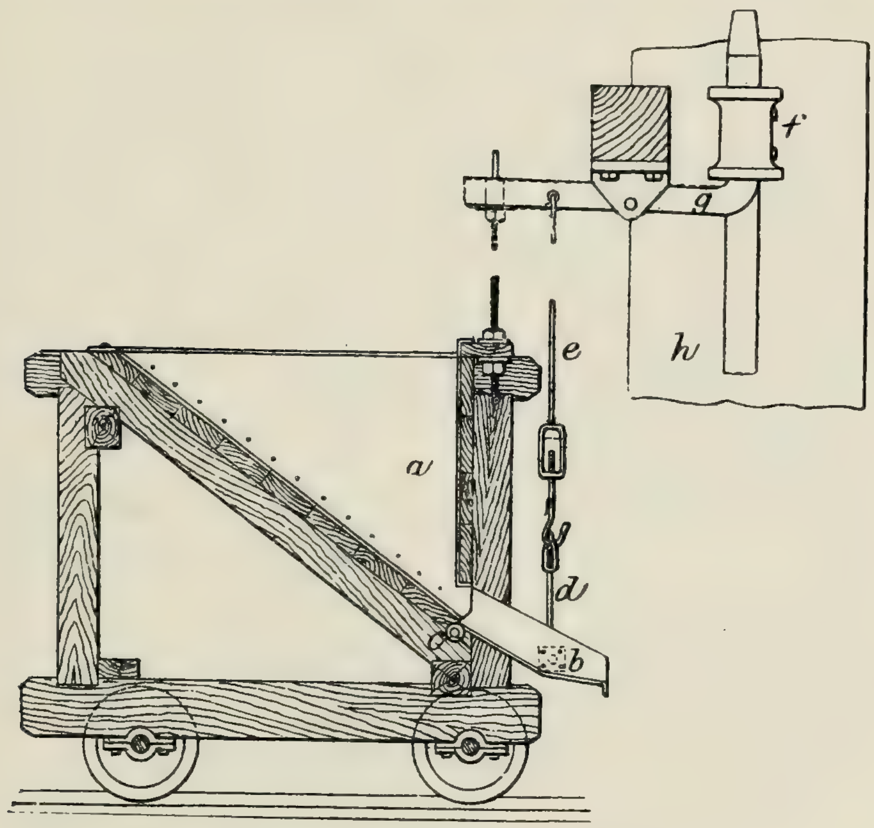



FIG. 53.—STANFORD FEEDER.

of ore. Springs or balance-weights are used to restore the shoot to its original position after each discharge. The angle of the shoot can be increased or reduced at the will of the millman, in accordance with the demands of the physical condition of the ore.

The same principle of positive action is applied to all automatic feeders. One of the simplest and cheapest is Stanford's, shown in Fig. 53. It consists of a hopper *a* with adjustable spout *b*, swung on trunnions *c*, and attached to a bar *d*, suspended from an adjustable rod *e*. A feeding-tappet *f* is keyed upon the battery-post *h*, and a lever *g* rotating on pivots is fixed to the rod *e*, so as to be struck by the

tappet, the lever *g* being forked that it may span the stem. While the battery is supplied with sufficient ore, the tappet does not descend far enough to encounter the end of the feeding-rod ; when the ore gets low, the tappet does strike the rod, and the effect is an oscillation of the front spout on its trunnions, whereby the ore is thrown forward. The machine being on wheels, it can be easily drawn back out of the way when necessary.

The Tulloch feeder, Fig. 54, is in very common use. Below the hopper, which will hold about $\frac{3}{4}$ t. of ore, is suspended a tray, and attached to this is a bumper-rod. The descending

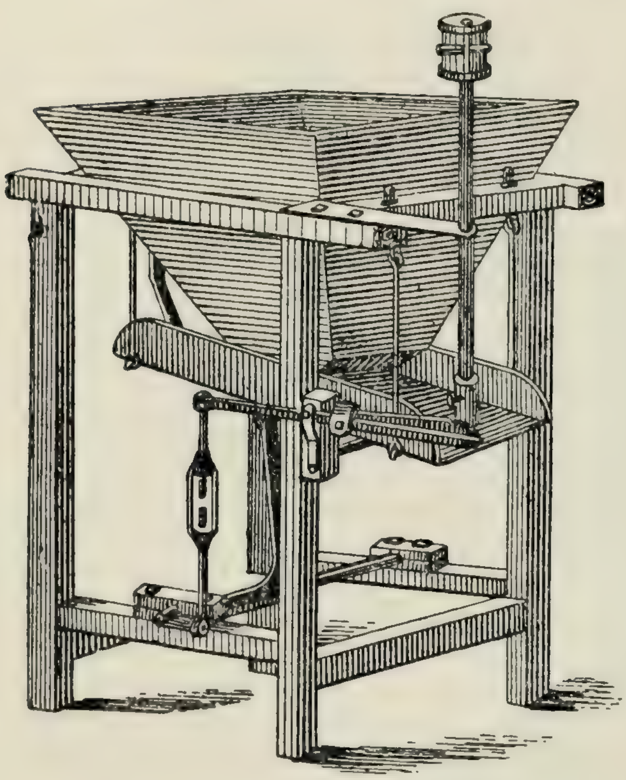


FIG. 54.—TULLOCH FEEDER.

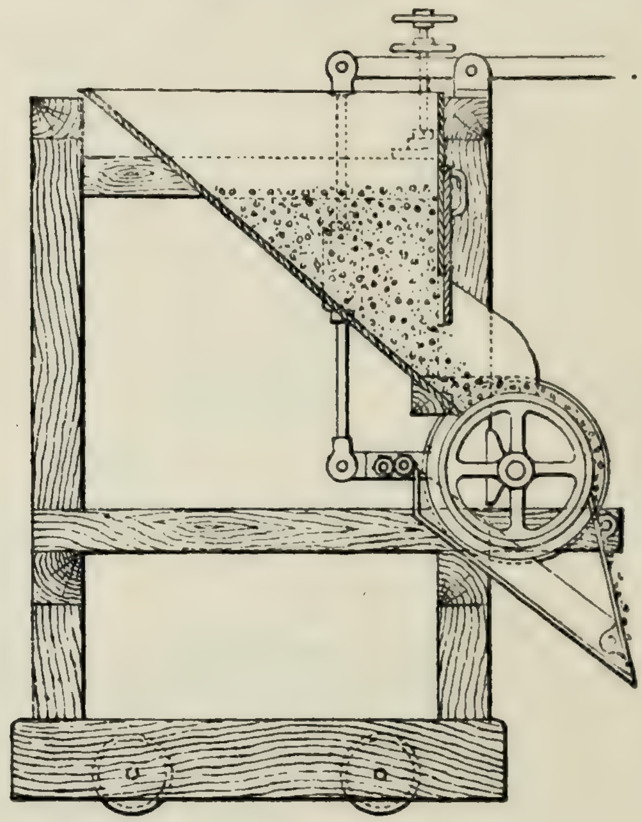


FIG. 55.—TEMPLETON FEEDER.

tappet of the central stamp, striking the bumper, imparts a backward movement to the tray, and a portion of the contents is scraped off by a small gate hinged to the hopper, and which rests at an angle on the bottom of the tray. It may be stationary or on wheels. The weight is about 700 lb., and cost about 26*l.*

The Templeton roller feeder, seen in Fig. 55, is not so well known, but with reasonably dry ores the author has found it to be a most satisfactory machine, uncostly, simple, reliable, and enduring—in fact, he prefers it to any. It is lighter and cheaper, and can be more easily repaired.

The Challenge feeder is made in two types, standard and suspended. The latter is shown in Fig. 56, and differs from the former only in being hung from timbers or rails attached to the battery-frame. The plate *a* below the hopper is set at an angle, and is rotated by bevel gear *b*, which is set in motion by a friction-grip varying according to the blow received from the tappet on the feed-stamp. At each partial rotation of the plate *a*, a certain amount of the ore is scraped off by the

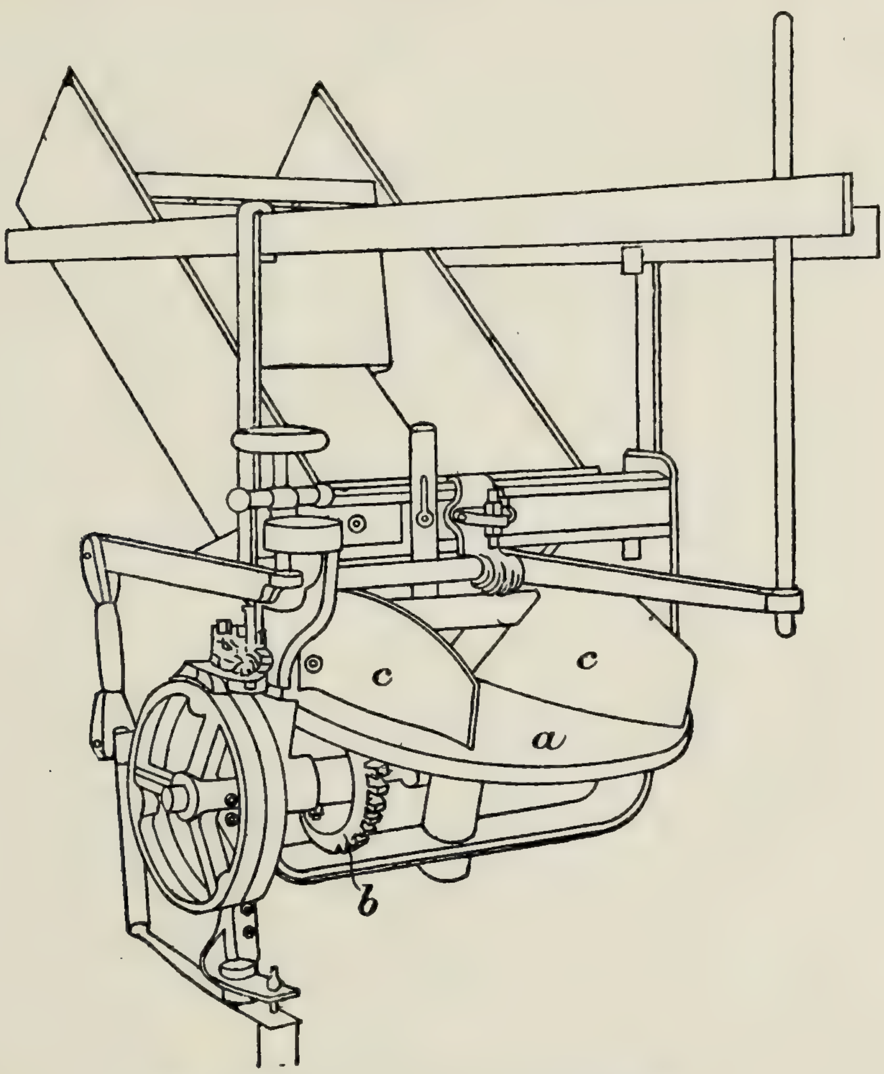



FIG. 56.—CHALLENGE FEEDER.

stationary wings *c*. The machine weighs 800 to 850 lb., and costs 30*l.* to 35*l.* It is popularly regarded as the most perfect feeder, especially with wet or sticky ores, and the suspended form is almost universal in the Rand batteries, and in the Alaskan mills. At the Alaska United, the plate is made of manganese steel instead of the usual cast iron. With hard, fine, gritty ore, trouble is apt to occur by packing between the plate and the wings. The mechanism is complicated, and troublesome to repair.

**Water.**—An abundant supply of good water is a prime necessity for the battery and its almost invariable accompaniment of concentrating appliances. Storage accommodation for 6 to 24 hours' consumption should be provided at a level appreciably above the top of the mortar boxes, so that the requisite supply will flow by gravitation and not be dependent upon the uninterrupted service of a pump or other mechanism. The distance between storage tank and battery should be as little as is convenient, to minimise cost of piping, risk of leakage, etc. The pipes should be protected both from heat and from cold. The distribution of the water is effected by leading a main of ample dimensions right through the battery at an elevation just above the top of the mortars. From this main, supplies are drawn into each mortar, either by a single inlet pipe, by a similar pipe furnished with perforated arms, or, as is perhaps best, by two service pipes, one towards each end of the box. This last plan helps to prevent the pulp packing beyond the first and fifth stamps.

A novel feature in a Canadian battery described by J. E. Hardmann * is an upward pointing jet (from the base of the chuck-block) between each two dies; it is said to prevent packing of sulphurets, and to aid amalgamation.

Almost (if not quite) the same contrivance is recommended by B. MacDonald,† who used it at the Dufferin mill, Nova Scotia. From a 3-in. main, receiving a supply under 30 ft. head, 6 ¾-in. pipes are passed in through the mortar, and deliver water about midway up the die level when new.

It is difficult to discover any real advantage in this method, while it possesses the practical drawback that pulp must find its way into the pipes and is liable to cause an obstruction whenever the water pressure becomes for any reason diminished to a point which is insufficient to maintain a clear passage. Fine pulp once admitted will pack tight and become very difficult of removal.

Every service pipe must be provided with an easily-regulated cock. It should be borne in mind that the pressure

* Jl. Fed. Can. Min. Inst., ii. 106 (1897).

† Trans. Can. Min. Inst., iii. 54.

diminishes towards the end of the main, so that the service cocks must be adjusted accordingly.

Sometimes a supplementary water spray is provided outside the box, to help carry off the pulp; but this should not be necessary when plates immediately follow the battery, because the full demands are better supplied inside the box.

Concentrators of all kinds require some additional water.

The quality of the water has to be considered. Water for battery purposes cannot be too clean. Solid impurities carried in suspension are objectionable, because they help to choke pipes and cocks, cause a syrupy tendency which interferes with plate amalgamation, and add to the slime difficulty in subsequent treatment of tailings. Muddy water should be settled and clarified, by passage through sand filter-beds if necessary. At many mills the supply is so deficient that water leaving the battery has to be used again and again, after opportunity has been given for deposition of suspended matter; but loss of gold often results from too little attention being given to making the clarification as complete as it might be.

Anything in the nature of grease must be rigidly excluded, for which reason lubricants must be kept out of the pulp, and water which has been used in condensing is generally excluded. The existence of grease may be combated by adding an alkali to the water, thus forming a soap. Quicklime, or a lye from wood ashes, may be used, but not the ashes themselves, as they contain much fine charcoal, which will be inimical to subsequent cyaniding of tailings. Treatment of greasy water is best done before admission to the battery, the scum formed being carefully removed. Grease of all kinds is highly prejudicial to amalgamation.

Impurities in solution in water are generally either saline or acid, and are due to various causes. Under exceptional circumstances, sea water has to be used, and losses of gold ensuing have been attributed to dissolution of the gold by the chlorides in the water. This, however, is contrary to the opinions of leading metallurgists, who assert that sea water has but a trifling solvent action on gold, and it seems much

more likely that the increased density of the water leads to a greater than usual loss of excessively fine gold in a state of suspension. In Western Australia the lack of fresh water is very marked. The water from shafts and wells about Kalgurli contains 5 to 8 % of soluble salts, largely consisting of magnesia. The Great Boulder mill water generally carries 15 % of sodium chloride besides magnesium salts. In summer time the water of Hannan's Lake contains 28 % total solids, about 25 being common salt, and the balance calcium and magnesium salts. Lake Austin holds the record with 30 % of saline impurities. An analysis of one mill water, taken after rain had materially diluted it, gave 4·3 % sodium chloride, ·8 % magnesium sulphate, and ·66 % calcium carbonate. With such water it is not surprising that a large proportion of the fine gold escapes arrest on amalgamating and concentrating surfaces.

Mine waters for the most part give an acid reaction. This is due to the presence of soluble sulphates arising from partial decomposition of pyrites in the ore. The generation of these sulphates is not confined to the action going on in the waters of the mine, but often proceeds while ore is being crushed in its passage through the battery, so that the remedy does not simply consist in avoiding mine water for battery work. Such acid water is very prejudicial to a high percentage of gold extraction, mercury being freely attacked by it and rendered inoperative. In addition, it causes extravagant wear of screens, etc. The evil can be entirely remedied by addition of quicklime : to the water in the supply tank, if it be that mine water is causing it, so that the deposited lime sulphate resulting can be periodically removed ; or to the ore while crushing, if it be to blame. Such a quantity as ¼ lb. of quicklime per ton of ore milled will usually suffice, and its cost is not worth considering. Moreover, it remains to exert further beneficial influence when cyanidation of tailings is to follow. The only objection to it is that, if used in great quantity, it may render the water unfit for use in boilers, and thus necessitate a separate source of supply for them ; that is to say, water which had once passed through the battery

would be used only for battery purposes. This, indeed, is a preferable arrangement in any case.

The volume of water supplied to the battery must be proportioned to the quantity of pulp produced, the fineness to which the crushing is carried, and the character of the stone under treatment, more especially if it be argillaceous, talcose, or highly pyritic. The pulp must be rendered thin enough to flow evenly and steadily over the amalgamating tables, and this condition will generally be compatible with a depth in the mortar of several inches above the lower edge of the screen. In fact, in some batteries the depth is such that the stamp is always immersed.

Double-discharge mortars require extra water to maintain a good issue, because in them the splash is weakened.

Additional water has always to be provided for the concentrating appliances which generally follow the battery operations, whether these be the old-fashioned blanket strake or some form of vanner.

WATER USED IN BATTERIES.

| — | Duty. | Screen. | Water used. | |
|---|---|---|---|---|
| | | | Per Stamp per hour. | Per Ton crushed. |
| California: | t. | gauge. | gal. | gal. |
| Clinton | 3·00 | .. | 300 | 2400 |
| Empire | 1·50 | 30 | 240 | 3840 |
| Gover | 2·75 | 20 to 50 | 195 to 325 | 1702 to 2836 |
| Idaho | 2·00 | 40 | 195 | 2340 |
| Kennedy | 2·50 | .. | 180 | 1728 |
| Keystone | 2·50 | .. | 195 | 1872 |
| North Star | 1·59 | 40 | 120 to 240 | 1811 to 3622 |
| South Spring Hill | 2·30 | .. | 210 | 2191 |
| Wildman | 2·30 | .. | 180 | 1878 |
| W. Y. O. D. | 1·70 | 40 | 180 | 2541 |
| Colorado: | | | | |
| Gregory Bobtail | 1·04 | 40 to 60 | 140 | 3230 |
| Hidden Treasure | 1·14 | 50 | 120 | 2526 |
| New York | 1·07 | 50 | 80 | 1794 |

WATER USED IN BATTERIES—*continued.*

| — | Duty. | Screen. | Water used. Per Stamp per hour. | Water used. Per Ton crushed. |
|---|---|---|---|---|
| | t. | gauge. | gal. | gal. |
| Prize | ·80 | 50 | 90 | 2700 |
| Randolph | ·93 | 50 | 85 | 2193 |
| Dakota: | | | | |
| Homestake | 4·00 | 30 | 190 | 1140 |
| New South Wales: | | | | |
| Lucknow | 1·37 | 30 | 120 | 2628 |
| New Zealand: | | | | |
| Phœnix | 1·57 | 140 | 240 | 3668 |
| Premier | 1·34 | 180 | 210 | 3761 |
| Reliance | 2·24 | 200 | 300 | 3214 |
| Transvaal: | | | | |
| Average | 4 to 5 | 600 to 900 | 240 to 390 | 2000 |
| Victoria: | | | | |
| Ararat | 1·25 to 1·5 | 90 to 120 | 180 to 540 | 3456 to 8640 |
| Ballarat | 1 to 4 | 40 to 200 | 40 to 360 | 960 to 2160 |
| Beechworth | ·8 to 4 | 60/140 | 30/480 | 900 to 2880 |
| Bendigo | 1 to 3·75 | 64/140 | 170/360 | 2304 to 4080 |
| Britannia | 2·35 | 120 | 300 | 3064 |
| Castlemaine | 1 to 3·25 | 40/144 | 200/540 | 3240 to 3987 |
| Catherine | 2·35 | 143 | 235 | 2400 |
| Dixon's | 3·69 | 100 | 600 | 2818 |
| Gippsland | 1·5 to 2 | 70/250 | 70/1000 | 1120 to 12,000 |
| Harrietville | 1·68 | 240 | 300 | 4286 |
| Hillsborough | 1·40 | 200 | 210 | 3600 |
| Maryborough | 1 to 3 | 70/144 | 40/360 | 960 to 2880 |
| New Normanby | 2·24 | 120 | 300 | 3214 |
| North Cornish | 2·01 | 160 | 150 | 1910 |
| Oriental | 1·45 | 220 | 240 | 3972 |
| Pearl | 2·46 | 168 | 390 | 3809 |
| Port Phillip | 3·36 | 81 | 360 | 2571 |
| Railway | 1·79 | 200 | 240 | 3218 |
| South Clunes | 2·80 | 100 | 480 | 4114 |
| South St. Mungo | 2·35 | 115 | 450 | 4591 |
| Star of the East | 2·24 | 200 | 450 | 4821 |
| Stephen | 1·68 | 250 | 270 | 3857 |

To what a wide extent the total consumption varies, may be judged from the table on pp. 159–160. Hardly any two mills agree, except where, as on the Rand, they are working under practically identical conditions. The low figure of the Homestake batteries is somewhat remarkable, with such a notably pyritic ore.

At the Wentworth mine, Lucknow, N.S.W., the author experienced difficulty in computing the consumption of water in the battery, because many boilers are also served by the same tank. This is placed near to and above the battery, is constructed of cement-lined concrete, and measures 61 ft. 5 in., by 40 ft. 4 in. by 7 ft. deep: if filled, it would hold over 60,000 gal., but an escape pipe has been placed at 4 ft. from the bottom, and the tank is seldom filled above this, on the ground of safety. It is fed by two Tangye pumps, one of 4-in. and one of 5-in. cylinder, each with 12-in. stroke. The former should deliver about 4800 and the latter 5000 gal. per hour, or approximately a total of 235,000 gal. per 24 hours, calculating on the displacement of the pump buckets at a piston speed of 75 ft. per minute, without deduction for loss by leakage, slip, etc. By rough observation, it was calculated that about half the total water consumption could be debited to the battery. This contains 30 stamps in constant work, with two Frue vanners per battery. Thus the figures work out at about 3600 gal. per head (including vanners) per 24 hours. At a duty of 1·37 t. per head per 24 hours, this means about 2628 gal. per ton, or more than $1\frac{1}{4}$ gal. of water per lb. of ore milled. Allowing for losses in various ways, notably in the wet concentrates taken from the vanner boxes, the proportion of water to solids in the escaping tailings pretty closely confirms these figures. The ore carries much lime (calcite), causing slimes, but part of the gangue is very heavy; concentrates, embracing coarse granular iron pyrites, and fine floury mispickel, average $2\frac{1}{2}$% by weight (dry) of the ore milled. After settlement, the water is used again and again, both in the battery and in the boilers, being quite free from sulphates and noxious minerals.

The volume of water supplied to the battery has consider-

able effect upon the fineness of the product, by reason of its influencing the facility of issue of the pulp. In some experiments made in New Zealand, with a 30-mesh screen and using only 1·6 t. of water to the ton of ore (that being of course very much below ordinary practice), it was found that 69·2 % of the pulp issuing would pass through 100 mesh. The same kind of ore was then stamped with a 60-mesh screen and 2·8 t. of water per ton of ore, when only 68 % would pass the 100-mesh screen. This clearly shows that by increasing the water 75 % the proportion of slimes made was actually lessened, though the fineness of the screen was increased 100 %. But it must be borne in mind that even 2·8 t. of water per ton of ore is so far beneath the figures of everyday usage, that results on the same scale cannot be looked for in ordinary wet milling. For instance, to augment the water supply from 7 t. to 14 t. per ton of ore would not produce corresponding results; on the contrary, after a certain point, augmenting the water would actually lessen the output of solids.

Though every precaution be taken to minimise waste or loss of water by settling tailings and re-using the clarified water, a certain consumption is inevitable from absorption by the ore. This is subject to great variation. On the Rand, it is computed that residual "sands" from cyaniding carry about 30 gal. of water per ton (12 to 14 % by weight), while "slimes" contain 240 gal. per ton (100 %). An American experimenter reckons 66 gal. per ton for hard quartz, and 96 gal. for soft ores.

Some recent trials made in the Mojave Desert, California, by A. W. Warwick * are most instructive. At first pointed boxes only were used as water-saving apparatus, but proved incapable of separating more than 45 %. With a siphon discharge but 20 % of the water could be saved, 80 % being necessary to prevent the effluent from clogging; with an adjustable gate, the difficulty was much reduced. Trouble was caused by coarse sands, so a sand-box was interposed, and then the pointed box effected a maximum economy

* En. & Min. Jl., May 5, 1900.

of about 80 %. In a month's run the figures obtained were :—

| Week. | Water supplied, gal. | Hours run. | Water per stamp per hour, gal. Using once. | Actually used. | Recovered, % |
|---|---|---|---|---|---|
| 1 | 13,154 | 82 | 80 | 16·0 | 80·0 |
| 2 | 16,760 | 80 | 95 | 22·2 | 76·8 |
| 3 | 15,487 | 81 | 93 | 19·1 | 79·4 |
| 4 | 14,850 | 80 | 92 | 18·5 | 79·8 |

The character of the ore crushed and the products from sand boxes and pointed boxes, together with percentages of water, are shown in the following table :—

| | From copper table. | Sand box. | Pointed box. |
|---|---|---|---|
| | % | % | % |
| Water in product . . . . . . | — | 29 8 | 50·1 |
| Screen mesh : | | | |
| Through 30 and on 40 . . . | 0·41 | 1·60 | — |
| Through 40 and on 80 . . . | 9·55 | 18·90 | — |
| Through 80 and on 100 . . . | 10·58 | 15·38 | 3·20 |
| Through 100 . . . . . . . . . | 78·75 | 63·35 | 96·30 |
| Total . . . . . . . . | 99·29 | 99·23 | 99·50 |

As nearly as could be estimated, the sand boxes saved four-fifths of the crushed rock and the pointed boxes one-fifth. The amount of rock crushed per 12-hour shift was $17\frac{3}{4}$ t., or 497 t. for the whole month.

During one month, careful measurements of water supplied to the mill were made. It was found that 80 gal. of water per stamp per hour were required by the most quartzose ore when the water was used but once. The most clayey ore required 95 gal. water. The ore consisted of a highly kaolinised porphyry, with but a small amount of pyrites, crushed to 30 mesh screen. The average amount of moisture in the tailings was 33·86 %.

M 2

The sand boxes used were 4 ft. wide by 10 ft. long by 3 ft. deep, and the pointed boxes were 6 ft. wide by 10 ft. long by 6 ft. 6 in. deep. It was found that, in order to settle the slimes thoroughly, the delivery of slime and water to the pointed boxes should not exceed 92½ gal. per foot of width per hour. A convenient figure for estimating may be taken as 100 gal. of slime per foot of width. Then two of the pointed boxes are required for a 10-stamp mill running continuously, the slime being divided between them. It is convenient, however, to have a third pointed box to provide for contingencies. A sand box of 4 ft. width and with any desired depth and length, should be provided for each pointed box.

**Driving.**—A consideration of the manner in which a battery shall be driven embraces many interesting problems.

*Power required.*—The various factors constituting the work to be done in a stamp mill, and the amount of power consumed by each have been calculated with great care by Prof. Louis,* who arrives at the conclusion that a 10-head battery of 900-lb. stamps making 90 drops of 7 in. (nominal) per minute will require 17·22 h.p. to be applied to the driving belt, or about 1¾ h.p. per stamp; this figure gives a good margin for safety. Morison found that the power required to run a 950-lb. stamp-battery at its maximum practical speed of 95 drops per minute was about 3 i.h.p., "and of this only about 1·7 h.p. was represented by weight lifted." The official returns of the Victorian Mines Department state the h.p. per stamp at figures ranging from ½ to 2½ h.p. The Homestake Co. reports its consumption of power at 1¾ h.p. per stamp on 900-lb. stamps dropping 8 to 9½ in. nominal at 85 to 90 times a minute. The North Star mill, California, employs water power, and 40 head of 850-lb. stamps, dropping 6 to 8 in. nominal, 82 to 85 times a minute, require 93 miners' inches (of 1·574 cub. ft.), per minute under a 277 ft. head, and a pressure of 212 to 215 lb. per sq. in., driving a 6 ft. Pelton wheel; at 75 % efficiency, this is equal to about 55 h.p. On

* 'Handbook of Gold Milling,' p. 220.

the same bases, the breakers absorb about 11 h.p. in addition, making a total of 66 h.p. for 40 stamps including breakers, or 1·65 h.p. per head.

It is usual to provide fully 2 h.p. per stamp, including breakers and concentrators.

*Steam Power.*—Steam is the most generally applicable form of power for mills, and a few facts concerning it may be useful.

Water which flows from a battery is rarely fit for boiler use. Even after most careful and complete settling, to remove suspended matter, it will usually contain in solution either acid sulphates from decomposition of pyrites in milling, or lime which has been added to neutralise such acids, or to facilitate settlement of slimes. Mine water is also often rendered harmful by saline or acid impurities.

Water consumption of boilers is equal to 1 cub. ft. (7½ gal.) per hour for each nominal h.p.

Scale may be prevented by introducing carbonate of soda (1 to 5 lb. a day) or caustic soda (½ to 2½ lb. a day) with the feed water, the quantity varying with size of boiler and degree of hardness of water. Kerosene is even more searching. Blowing off frequently is also a potent check, especially when the boilers have cooled.

Fuel consumption is ordinarily about 7½ lb. coal or 15 lb. dry wood per cub. ft. of water evaporated, or about 1 lb. coal (2 lb. wood) per gallon. Each sq. ft. of grate area, under natural draught, will burn 10 to 12 lb. hard coal or 18 to 20 lb. soft coal per hour.

Relative heating values can only be approximated, because of variations in quality and condition.

Freshly won coal is much superior to that which has been long mined ; after 6 months' exposure to air, coal deteriorates rapidly.

Wood is very wet when green, and becomes valueless if kept too long.

In recent experiments, 1 lb. anthracite coal evaporated 9·7 lb. water; 1 lb. bituminous coal, 10·14 lb. ; and 1 lb. fuel oil of 36°, 16·48 lb. ; while D. K. Clark gives the theoretical

heat values, in British thermal units, of 1 lb. good coal as 14,700; 1 lb. dry wood, 10,974; and 1 lb. petroleum, 20,411; and Urquhart's experiments showed Russian light crude oil, 22,027; Pennsylvania heavy crude, 20,736; Russian heavy crude, 20,138; petroleum refuse, 19,832; good English coal, 14,112.

Tests on well dried white oak wood indicate that 1 cord (128 cub. ft.) weighing 3850 lb., is of the same heating value as 1550 to 1700 lb. bituminous coal.

The mean deduction from these statements would be that 1 ton of good coal is about equal to $4\frac{3}{4}$ barrels (of 42 gal.) of petroleum, or $\frac{3}{4}$ cord of dry oak wood.

White oak is almost twice as good as pine, elm, or chestnut. The white box of Australia is superior to oak. In the Cape, small mimosa wood or "thorn," costing 20 to 25*s.* per long ton, is much used, at the rate of about 7 lb. per h.p. per hour. But much depends upon having the firebox suited to the fuel.

Where a very inferior fuel, such as culm, slack, coal-dust or coke-dust can be had, its low price makes it most economical when burned in a suitable furnace, such as the Meldrum. The following figures are quoted from Bryan Donkin :—

COST OF FUELS.

| Kind of Fuel. | Price per ton. | Water evaporated per lb. of fuel. | Cost of fuel per 1000 gal. evaporated. |
|---|---|---|---|
| | *s.* | lb. | *s. d.* |
| Large Welsh coal . . . . . | 22 | 9 | 10 11 |
| Dust do. . . . . . | 10 | 8 | 5 3 |
| Dust coke . . . . . . . . | 5 | 6 | 3 8 |

Steam pipes should be laid in such a manner that no transverse strain comes upon the joints to cause leakage, and should be most efficiently lagged with felt or boxed in sawdust to reduce loss of heat by radiation to a minimum. Discretion must also be used in choice of size, bearing in mind

that too small pipes create excessive friction, and that too large ones expose an extensive surface to radiation.

Selection of an engine must often depend on the class of labour available for driving it. The simplest will in many cases prove most suitable in the end. But the relative running costs of the principal kinds, deduced from calculations by Dr. Emery, on a basis of 500 net h.p. delivered for 10 hours a day and 308 days in the year, supposing a simple engine operating continuously at its normal capacity, are :—

RELATIVE COST OF ENGINE POWER.

| Kind of Engine. | Coal at 8/4 p. ton. | | Coal at 12/6 p. ton. | | Coal at 15/8 p. ton. | |
|---|---|---|---|---|---|---|
| | Cost per h.p. per ann. | Cost per h.p. per hour. | Cost per h.p. per ann. | Cost per h.p. per hour. | Cost per h.p. per ann. | Cost per h.p. per hour. |
| | *s.* *d.* | *d.* | *s.* *d.* | *d.* | *s.* *d.* | *d.* |
| Simple, high speed . . | 124 2 | ·483 | 150 8 | ·587 | 177 3 | ·690 |
| Simple, low speed . . | 118 7 | ·462 | 142 6 | ·555 | 166 5 | ·648 |
| Simple, low speed, condensing . . . . . | 95 1 | ·370 | 111 6 | ·434 | 128 0 | ·498 |
| Compound, low speed, condensing . . . . | 93 1 | ·362 | 106 4 | ·414 | 121 2 | ·472 |
| Triple expansion, low speed, condensing . . | 91 6 | ·356 | 105 6 | ·411 | 117 10 | ·459 |

These are theoretical figures and are never attainable in practice. They must be increased by at least 50 to 75 % all through, and with small engines under the best practicable conditions they may be doubled.

The governor of a battery engine should always be of the type which automatically cuts off steam when disconnected by the breaking of the belt; a runaway engine can wreck a battery in a few moments.

*Water power.*—Water power is by far the cheapest when it is constant and reliable. The following table, calculated on an available efficiency of 75 %, gives the quantity of water

in cub. ft. per minute) for certain powers under various heads.

HORSE-POWER OF WATER.

| Fall. | 8 | 10 | 15 | 20 | 30 | 40 | 50 | 60 | 70 | 80 |
|---|---|---|---|---|---|---|---|---|---|---|
| ft. | | | | | | | | | | |
| 3 | 1883 | 2353 | 3530 | | | | | | | |
| 4 | 1412 | 1765 | 2648 | 3530 | | | | | | |
| 6 | 941 | 1176 | 1765 | 2353 | 3530 | | | | | |
| 8 | 706 | 883 | 1324 | 1765 | 2648 | 3530 | | | | |
| 10 | 565 | 706 | 1059 | 1412 | 2118 | 2824 | 3530 | | | |
| 12 | 471 | 588 | 883 | 1176 | 1765 | 2353 | 2940 | 3530 | | |
| 15 | 377 | 471 | 706 | 942 | 1412 | 1884 | 2353 | 2824 | 3295 | |
| 20 | 282 | 353 | 530 | 706 | 1059 | 1412 | 1765 | 2118 | 2471 | 2824 |
| 25 | 226 | 282 | 424 | 565 | 847 | 1130 | 1412 | 1694 | 1977 | 2260 |
| 30 | 189 | 236 | 353 | 471 | 706 | 942 | 1176 | 1412 | 1648 | 1883 |
| 35 | 161 | 202 | 303 | 403 | 606 | 806 | 1010 | 1212 | 1412 | 1612 |
| 40 | 141 | 176 | 265 | 353 | 530 | 706 | 883 | 1059 | 1235 | 1412 |
| 45 | 125 | 157 | 235 | 314 | 471 | 628 | 784 | 941 | 1098 | 1255 |
| 50 | 113 | 141 | 212 | 282 | 423 | 565 | 706 | 847 | 988 | 1130 |
| 60 | 94 | 118 | 176 | 235 | 353 | 471 | 588 | 706 | 824 | 942 |
| 70 | 81 | 101 | 151 | 202 | 303 | 403 | 505 | 606 | 706 | 807 |
| 80 | 71 | 88 | 132 | 176 | 265 | 353 | 441 | 530 | 618 | 706 |
| 100 | 56 | 71 | 106 | 141 | 212 | 282 | 353 | 424 | 494 | 565 |

*Gas Engines.*—Theoretically the gas engine is capable of producing 1 i.h.p. by a consumption of ·8 lb. coal, which is approximately 3 times the efficiency of the best designed and constructed steam engine. If in practice it does not absolutely sustain this high rate, it is always much superior to the steam engine on this ground, and possesses moreover many other features which enhance its usefulness. The first cost of a gas-engine plant, including the producer, is much less than that of engines and boilers generating an equivalent steam power. Then again, while large size is a great factor in economy with the latter, and therefore a strong incentive to centralisation of generating plant, the small gas engine is relatively as efficient as the large, so that nothing is lost by a

multiplication of the motors, thus often saving long transmissions of steam or additional intermediate driving gear. The consumption of water is a very small matter indeed, being only that necessary for cooling the cylinder, which, with scarcely any loss, may be used over and over again; the quality, too, is of little importance, so long as it is not highly corrosive. Where liquid fuel (mineral oil) or heated gases from coke ovens or blast furnaces may be had, the gas engine needs no producer, and can be run at exceedingly low cost. The producer will make good use of any fuel, however inferior.

Some working results at a "desert mill" in California are published by A. W. Warwick,* who used a Fairbanks-Morse engine of 34 brake h.p. to run 10 stamps (850 lb., 6 in. drop, 90 per min.), breaker (7 × 10 in.), elevator (25 ft.), and 2 vanners (6 ft.). The fuel consumed is *puente* distillate of 45°, costing 5¼*d.* per gal. at the mill, which works out at ·085 gal. per h.p. per hour, equal to ·87*d.* Labour required is almost nil; and loss of cooling water (by evaporation) is 55 gal. per 12-hour shift.

*Electric Power.*—Occasionally it may be possible to select central electric stations, capable of providing the power for a number of motors for driving various stamp batteries in the district. Where this can be done, the electric motor is decidedly the most suitable form of driving power to employ; but instead of making use of continuous-current machinery, it is clearly a case for the application of polyphase machinery, which requires considerably less attention and up-keep than machinery of the continuous-current type. The main points in which multiphase machinery differs from continuous-current machinery are probably well known to most engineers, the multiphase motors being entirely devoid of brushes or commutators, which have been found to be the chief source of trouble with the continuous-current motor. This system is particularly suitable for mining work, as the pressure can be transformed up or down at will, and the power conveyed for enormous distances with the least possible capital outlay in copper mains.

* En. & Min. Jl., May 5, 1900.

*Line Shafting.*—It has already been mentioned (p. 57), when dealing with the battery framing, that various situations are chosen for the line shafting. Perhaps the most usual system is to carry it on the frame which supports the cam floor, so that the belting lies almost horizontally between line-shaft and cam-shaft. The drawbacks of this method have been described (p. 59).

Another simple plan is to carry the line-shaft right through the centre of the mill, when there is a double row of batteries back to back, resting it on the ground sills, and taking driving belts from it on either side direct to the cam-shafts. This economises shafting, pulleys, and belting, but is objected to on the score that the main-shaft bearings are out of sight, and in a dusty position.

More complicated methods of arrangement are a line-shaft as before on the ground sills, with a counter-shaft in addition placed under the feed floor behind the batteries. This seems a needless multiplication of gear without any benefit, and makes such short centres that tighteners are a necessity.

The best position for the motor is midway in the length of line shafting, so that the load may be equalised. The shafting may be somewhat reduced in diameter as it gets farther away from the motor.

For computing the h.p. of shafting, a safe formula for prime motor or main shafting, with pulley placed not more than 1 ft. from main bearing, is:

$$\frac{\text{diam}^3 \times \text{rev. per min.}}{100} = \text{h.p.}$$

For second motor or line shafting:

$$\frac{\text{diam.}^3 \times \text{rev. per min.}}{80} = \text{h.p.}$$

Dimensions (with wide safety margin) adapted to well-supported shafting carrying medium-sized pulleys are given in the table on p. 171; with good hammered-iron shafting of 4 in. diam. and upwards, some 50 % more load may be added,

but prime-motor shafting and that supporting cog-gear or very heavy pulleys should be increased by ¼ to 1 in.

DIMENSIONS OF SHAFTING.

| h.p. | Revolutions per minute. | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 30 | 40 | 50 | 60 | 80 | 100 | 125 | 150 | 175 | 200 | 250 | 300 |
| 1 | 1 5/16 | 1 1/4 | 1 3/16 | 1 1/16 | 15/16 | 7/8 | 13/16 | 3/4 | 3/4 | 11/16 | 5/8 | 1/2 |
| 2 | 1 11/16 | 1 1/2 | 1 7/16 | 1 5/16 | 1 3/16 | 1 1/8 | 1 1/16 | 1 | 15/16 | 7/8 | 13/16 | 3/4 |
| 3 | 1 15/16 | 1 3/4 | 1 5/8 | 1 1/2 | 1 3/8 | 1 5/16 | 1 3/16 | 1 3/16 | 1 1/8 | 1 1/16 | 1 | 1 |
| 4 | 2 1/8 | 1 15/16 | 1 13/16 | 1 5/8 | 1 1/2 | 1 7/16 | 1 5/16 | 1 1/4 | 1 3/16 | 1 3/16 | 1 1/8 | 1 1/16 |
| 5 | 2 1/4 | 2 1/8 | 1 15/16 | 1 13/16 | 1 5/8 | 1 1/2 | 1 3/8 | 1 5/16 | 1 1/4 | 1 1/4 | 1 3/16 | 1 1/8 |
| 6 | 2 7/16 | 2 3/16 | 2 | 1 7/8 | 1 3/4 | 1 5/8 | 1 1/2 | 1 3/8 | 1 5/16 | 1 1/4 | 1 3/16 | 1 3/16 |
| 7 | 2 9/16 | 2 1/4 | 2 1/8 | 2 | 1 7/8 | 1 13/16 | 1 9/16 | 1 1/2 | 1 3/8 | 1 5/16 | 1 1/4 | 1 3/16 |
| 8 | 2 5/8 | 2 7/16 | 2 1/4 | 2 1/8 | 1 15/16 | 1 13/16 | 1 5/8 | 1 5/8 | 1 1/2 | 1 7/16 | 1 5/16 | 1 1/4 |
| 9 | 2 3/4 | 2 1/2 | 2 5/16 | 2 3/16 | 2 | 1 7/8 | 1 3/4 | 1 5/8 | 1 1/2 | 1 1/2 | 1 3/8 | 1 5/16 |
| 10 | 2 7/8 | 2 5/8 | 2 7/16 | 2 1/4 | 2 1/16 | 1 15/16 | 1 13/16 | 1 11/16 | 1 9/16 | 1 1/2 | 1 7/16 | 1 5/16 |
| 11 | 2 15/16 | 2 5/8 | 2 1/2 | 2 5/16 | 2 1/8 | 2 | 1 13/16 | 1 11/16 | 1 5/8 | 1 1/2 | 1 7/16 | 1 3/8 |
| 12 | 3 1/8 | 2 3/4 | 2 9/16 | 2 3/8 | 2 3/16 | 2 1/16 | 1 7/8 | 1 13/16 | 1 11/16 | 1 5/8 | 1 1/2 | 1 7/16 |
| 13 | 3 3/16 | 2 7/8 | 2 5/8 | 2 7/16 | 2 1/4 | 2 1/8 | 1 15/16 | 1 7/8 | 1 11/16 | 1 5/8 | 1 9/16 | 1 7/16 |
| 14 | 3 3/16 | 2 15/16 | 2 11/16 | 2 9/16 | 2 5/16 | 2 1/8 | 2 | 1 7/8 | 1 13/16 | 1 11/16 | 1 9/16 | 1 1/2 |
| 15 | 3 3/8 | 2 15/16 | 2 3/4 | 2 5/8 | 2 5/8 | 2 3/16 | 2 1/16 | 1 15/16 | 1 13/16 | 1 3/4 | 1 5/8 | 1 1/2 |
| 16 | 3 3/8 | 3 1/8 | 2 13/16 | 2 5/8 | 2 3/8 | 2 1/4 | 2 1/8 | 1 15/16 | 1 7/8 | 1 13/16 | 1 5/8 | 1 9/16 |
| 17 | 3 7/16 | 3 3/16 | 2 7/8 | 2 11/16 | 2 7/16 | 2 5/16 | 2 1/8 | 2 | 1 7/8 | 1 13/16 | 1 11/16 | 1 5/8 |
| 18 | 3 1/2 | 3 3/16 | 2 15/16 | 2 3/4 | 2 9/16 | 2 5/16 | 2 1/8 | 2 1/16 | 1 15/16 | 1 7/8 | 1 3/4 | 1 5/8 |
| 20 | 3 5/8 | 3 1/4 | 3 1/16 | 2 13/16 | 2 5/8 | 2 7/16 | 2 1/4 | 2 1/8 | 2 | 1 15/16 | 1 13/16 | 1 11/16 |
| 25 | 3 13/16 | 3 1/2 | 3 5/16 | 3 1/8 | 2 7/8 | 2 5/8 | 2 7/16 | 2 5/16 | 2 1/8 | 2 1/8 | 1 15/16 | 1 13/16 |
| 30 | 4 1/8 | 3 3/4 | 3 1/2 | 3 1/4 | 3 | 2 7/8 | 2 9/16 | 2 7/16 | 2 5/16 | 2 3/16 | 2 1/16 | 1 15/16 |
| 35 | 4 5/16 | 3 15/16 | 3 5/8 | 3 7/16 | 3 3/16 | 2 15/16 | 2 11/16 | 2 9/16 | 2 7/16 | 2 5/16 | 2 1/8 | 2 1/16 |
| 40 | 4 9/16 | 4 1/8 | 3 7/8 | 3 1/2 | 3 5/16 | 3 3/16 | 2 15/16 | 2 11/16 | 2 9/16 | 2 7/16 | 2 1/4 | 2 1/8 |
| 45 | 4 7/8 | 4 1/4 | 3 15/16 | 3 3/4 | 3 7/16 | 3 3/16 | 2 15/16 | 2 3/4 | 2 5/8 | 2 1/2 | 2 3/8 | 2 3/16 |
| 50 | 4 15/16 | 4 7/16 | 4 1/16 | 3 13/16 | 3 1/2 | 3 5/16 | 3 1/8 | 2 15/16 | 2 11/16 | 2 5/8 | 2 7/16 | 2 1/4 |
| 55 | 5 1/8 | 4 5/8 | 4 1/4 | 4 | 3 11/16 | 3 3/8 | 3 3/16 | 3 | 2 7/8 | 2 11/16 | 2 1/2 | 2 3/8 |
| 60 | 5 1/4 | 4 3/4 | 4 3/8 | 4 1/8 | 3 3/4 | 3 1/2 | 3 1/4 | 3 1/8 | 2 7/8 | 2 3/4 | 2 9/16 | 2 7/16 |
| 70 | 5 1/2 | 5 | 4 9/16 | 4 3/8 | 3 15/16 | 3 5/8 | 3 7/16 | 3 1/4 | 3 1/8 | 2 7/8 | 2 11/16 | 2 9/16 |
| 80 | 5 3/4 | 5 1/4 | 4 15/16 | 4 1/2 | 4 1/16 | 3 13/16 | 3 9/16 | 3 7/16 | 3 1/4 | 3 1/8 | 2 7/8 | 2 11/16 |
| 90 | 6 | 5 7/16 | 5 1/8 | 4 3/4 | 4 1/4 | 4 | 3 3/4 | 3 1/2 | 3 5/16 | 3 3/16 | 2 15/16 | 2 3/4 |
| 100 | 6 1/4 | 5 5/8 | 5 3/8 | 4 15/16 | 4 7/16 | 4 1/4 | 3 7/8 | 3 11/16 | 3 3/8 | 3 1/4 | 3 1/8 | 2 7/8 |

Dimensions of Shafting—*continued.*

| h.p. | Revolutions per minute. | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 30 | 40 | 50 | 60 | 80 | 100 | 125 | 150 | 175 | 200 | 250 | 300 |
| 120 | $6\frac{3}{4}$ | 6 | $5\frac{11}{16}$ | $5\frac{1}{4}$ | $4\frac{7}{8}$ | $4\frac{7}{16}$ | $4\frac{1}{8}$ | $3\frac{13}{16}$ | $3\frac{5}{8}$ | $3\frac{1}{2}$ | $3\frac{1}{4}$ | $3\frac{1}{8}$ |
| 140 | $6\frac{15}{16}$ | $6\frac{3}{8}$ | $5\frac{13}{16}$ | $5\frac{1}{2}$ | 5 | $4\frac{5}{8}$ | $4\frac{1}{4}$ | 4 | $3\frac{13}{16}$ | $3\frac{11}{16}$ | $3\frac{7}{16}$ | $3\frac{1}{4}$ |
| 160 | $7\frac{1}{4}$ | $6\frac{3}{4}$ | $6\frac{1}{8}$ | $5\frac{3}{4}$ | $5\frac{1}{4}$ | $4\frac{15}{16}$ | $4\frac{7}{16}$ | $4\frac{1}{4}$ | 4 | $3\frac{13}{16}$ | $3\frac{9}{16}$ | $3\frac{5}{16}$ |
| 180 | $7\frac{1}{2}$ | $6\frac{7}{8}$ | $6\frac{1}{4}$ | $5\frac{15}{16}$ | $5\frac{7}{16}$ | $5\frac{1}{8}$ | $4\frac{3}{4}$ | $4\frac{1}{2}$ | $4\frac{1}{4}$ | 4 | $3\frac{3}{4}$ | $3\frac{1}{2}$ |
| 200 | $7\frac{13}{16}$ | $7\frac{1}{16}$ | $6\frac{11}{16}$ | $6\frac{1}{8}$ | $5\frac{3}{4}$ | $5\frac{3}{8}$ | $4\frac{15}{16}$ | $4\frac{5}{8}$ | $4\frac{1}{2}$ | $4\frac{1}{4}$ | $3\frac{7}{8}$ | $3\frac{5}{8}$ |
| 250 | $8\frac{1}{2}$ | $7\frac{1}{2}$ | $7\frac{1}{8}$ | $6\frac{11}{16}$ | $6\frac{1}{16}$ | $5\frac{3}{4}$ | $5\frac{1}{4}$ | 5 | $4\frac{5}{8}$ | $4\frac{7}{16}$ | $4\frac{1}{8}$ | $3\frac{13}{16}$ |
| 300 | 9 | $8\frac{1}{4}$ | $7\frac{1}{2}$ | $7\frac{1}{16}$ | $6\frac{7}{16}$ | 6 | $5\frac{1}{2}$ | $5\frac{1}{4}$ | 5 | $4\frac{3}{4}$ | $4\frac{7}{16}$ | $4\frac{1}{8}$ |
| 350 | $9\frac{3}{8}$ | $8\frac{1}{2}$ | $7\frac{15}{16}$ | $7\frac{3}{16}$ | $6\frac{7}{8}$ | $6\frac{7}{16}$ | $5\frac{3}{4}$ | $5\frac{7}{16}$ | $5\frac{5}{16}$ | 5 | $4\frac{5}{8}$ | $4\frac{3}{8}$ |
| 400 | $9\frac{7}{8}$ | 9 | $8\frac{3}{8}$ | $7\frac{1}{2}$ | $7\frac{1}{8}$ | $6\frac{3}{4}$ | $6\frac{1}{8}$ | $5\frac{3}{4}$ | $5\frac{7}{16}$ | $5\frac{1}{4}$ | $4\frac{7}{8}$ | $4\frac{1}{2}$ |
| 450 | $10\frac{1}{8}$ | $9\frac{1}{4}$ | $8\frac{3}{4}$ | $8\frac{3}{16}$ | $7\frac{3}{8}$ | $6\frac{15}{16}$ | $6\frac{1}{4}$ | 6 | $5\frac{3}{4}$ | $5\frac{3}{8}$ | $5\frac{1}{16}$ | $4\frac{5}{8}$ |
| 500 | $10\frac{1}{2}$ | $9\frac{5}{8}$ | 9 | $8\frac{3}{8}$ | $7\frac{1}{2}$ | $7\frac{1}{8}$ | $6\frac{3}{4}$ | $6\frac{3}{16}$ | $5\frac{3}{4}$ | $5\frac{9}{16}$ | $5\frac{1}{4}$ | $4\frac{7}{8}$ |
| 550 | $10\frac{7}{8}$ | 10 | $9\frac{1}{4}$ | $8\frac{7}{8}$ | 8 | $7\frac{5}{8}$ | $6\frac{3}{4}$ | $6\frac{1}{4}$ | 6 | $5\frac{3}{4}$ | $5\frac{3}{8}$ | 5 |
| 600 | $11\frac{3}{4}$ | $10\frac{1}{4}$ | $9\frac{1}{2}$ | $8\frac{15}{16}$ | $8\frac{1}{8}$ | $7\frac{3}{4}$ | $7\frac{1}{8}$ | $6\frac{3}{4}$ | $6\frac{1}{4}$ | $5\frac{15}{16}$ | $5\frac{9}{16}$ | $5\frac{1}{4}$ |

*Pulleys.*—The driving pulleys on the line shafting are of the usual cast-iron type, but iron or steel pulleys are quite out of place on the cam-shafts, because the vibration soon destroys them. For this latter situation, the built-up wooden pulley is always to be preferred, notwithstanding its excessive weight. A perfectly true face can be got by "turning" it up in place. Provision must be made for throwing each battery out of gear without interference with the others.

The following rules for calculating speeds may be useful:—

(1) Given diameter of driven pulley, to find its speed—multiply diameter of driver (in inches) by number of its revolutions per minute, and divide the product by diameter of driven.

(2) Given diameter and speed of driver, to find diameter of driven that shall make any given number of revolutions in the same time—multiply diameter of driver by its speed, and divide the product by speed of driven.

(3) To find size of driver necessary—multiply diameter of

driven by speed desired, and divide the product by speed of driver.

*Belting.*—Cotton rope is best for driving the line shafting, and leather belts for the cam-shafts. The upper side of a pulley should always carry the slack belt. The adhesion of belts is greater on polished than on rough pulleys, and much greater on a leather covered pulley than on polished iron. Leather belts require an occasional application of tallow, neats'-foot oil, or cod-liver oil, with a little rosin when they become hard and dried. Rubber belts must never be touched with any animal fat, but a little boiled linseed oil may be used if they slip.

The table of dimensions for belts given below is applicable to single leather, 4-ply rubber, and 4-ply cotton belting ; with double leather, 6-ply rubber or 6-ply cotton, transmission of 50 to 75 % more power can be safely undertaken.

DIMENSIONS OF BELTING.

| Speed, ft. per min. | Width of Belt. | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 2 in. | 3 in. | 4 in. | 5 in. | 6 in. | 8 in. | 10 in. | 12 in. | 14 in. | 16 in. | 18 in | 20 in. |
| | h.p. | h p. | h.p. | h.p. | h.p. | h.p. | h.p. | h.p. | h.p. | h.p. | h.p. | h.p. |
| 400 | 1 | 1½ | 2 | 2½ | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 |
| 600 | 1½ | 2¼ | 3 | 3¾ | 4½ | 6 | 7½ | 9 | 10½ | 12 | 13½ | 15 |
| 800 | 2 | 3 | 4 | 5 | 6 | 8 | 10 | 12 | 14 | 16 | 18 | 20 |
| 1000 | 2½ | 3¾ | 5 | 6¼ | 7½ | 10 | 12½ | 15 | 17½ | 20 | 22½ | 25 |
| 1200 | 3 | 4½ | 6 | 7½ | 9 | 12 | 15 | 18 | 21 | 24 | 27 | 30 |
| 1500 | 3¾ | 5¾ | 7½ | 9½ | 11½ | 15 | 18¾ | 22½ | 26½ | 30 | 33½ | 37½ |
| 1800 | 4½ | 6¾ | 9 | 11¼ | 13½ | 18 | 22½ | 27 | 31½ | 36 | 40½ | 45 |
| 2000 | 5 | 7½ | 10 | 12½ | 15 | 20 | 25 | 30 | 35 | 40 | 45 | 50 |
| 2400 | 6 | 9 | 12 | 15 | 18 | 24 | 30 | 36 | 42 | 48 | 54 | 60 |
| 2800 | 7 | 10½ | 14 | 17½ | 21 | 28 | 35 | 42 | 49 | 56 | 63 | 70 |
| 3000 | 7½ | 11¼ | 15 | 18¾ | 22½ | 30 | 37½ | 45 | 52½ | 60 | 67½ | 75 |
| 3500 | 8¾ | 13 | 17½ | 22 | 26 | 35 | 44 | 52½ | 61 | 70 | 79 | 88 |
| 4000 | 10 | 15 | 20 | 25 | 30 | 40 | 50 | 60 | 70 | 80 | 90 | 100 |
| 4500 | 11¼ | 17 | 22½ | 28 | 34 | 45 | 57 | 69 | 78 | 90 | 102 | 114 |
| 5000 | 12½ | 19 | 25 | 31 | 37½ | 50 | 62½ | 75 | 87½ | 100 | 112 | 125 |

*Lubrication.*—Inasmuch as grease or oil of any kind is most objectionable among the contents of the mortar-box, interfering seriously with the process of amalgamation, no substance of that kind should be used where it can be avoided about the battery. Cams and guides can be most effectively lubricated with soft-soap, into which a little finely-powdered graphite ("plumbago") may be stirred. Drip trays should be placed where they will catch any excess of lubricant that may escape from bearings.

**Crawls.**—The hoisting of stamps for the purpose of renewing shoes, bosses, tappets and stems is a very heavy and cumbersome job, unless provision is made for it by fixing a crawl or traveller well above the battery, either on the top of the king-posts of the frame, or suspended from the roof-timbers. The rail may be single or double, so long as it is strong enough. Light chain blocks are hung in the hook of the crawl, and connect at the other end with a simple ring having an eye attached to it. This ring, made of 1 to $1\frac{1}{4}$-in. square iron, is merely slipped a few inches down the stem, and the chain pulling on the eye at one side makes it pinch sufficiently tight to hold : an upward tap with a hand-hammer releases the stamp and allows it to fall forcibly when required. A crawl may be considered an absolute essential to a battery.

**Battery Floor.**—The floor of a battery should always be made with such a contour and of such material that it can be sluiced down into a gutter or drain leading to a trap of some gallons' capacity, from which the excess water can be drawn off into the tailings launder. The spilling of mercury, splashing of pulp, and slopping of water are quite impossible of prevention at all times, and, unless proper precautions are taken, there will be considerable losses of value. Nothing can beat a concrete floor with fine cement face. The solids collected in the trap (minus chips and other floating matter) should be periodically fed into one of the mortar boxes.

**Illumination.**—The lighting of a stamp mill is a matter of primary moment, though frequently it is utterly neglected,

and the men have to work in semi-darkness. No form of economy is more prejudicial to good milling. The light should be thrown directly on to the screens and plates, and on to the cam-shaft. Lamps, whether kerosene or electric, must be suspended, and special provision (by wire coil springs) must be made to take up vibration. The electric light is preferable, being always clean, and occupying very little room. Besides the fixed lights, a most useful addition is a movable 16-candle lamp attached to each battery, to be used where required.

**Bins.**—For each 10 stamps a bin should be provided with capacity sufficient to hold 1 to 6 days' supply of milling dirt. Bins are built of stout framing and 2-in. plank, and lined throughout with ¼-in. sheet-iron screwed to the planking. Usually the floor is made inclined towards the feed opening in front, as seen at *d* in Fig. 11 and in Fig. 22; but in many of the Rand mills rectangular bins are preferred, because they furnish accommodation for twice as much ore. In this latter case, the floor is flat, and the ore forms its own slope, which reduces the wear and tear on linings. Though shovelling is necessitated in order to take advantage of the extra accumulation, that is considered a small drawback compared with the advantage of having such a useful reserve in case of a hindrance to the current supply. A discharge door and shoot leading to each self-feeder hopper are part of the equipment, and resemble those already described on p. 8. When the breaker does not discharge into the bins, a car track (generally double) is laid along the tops of the bins.

**Weighing.**—It is quite a simple matter to fix a small weighbridge in the track to the breaker or in that to the battery bin, and thus to weigh every car-load of ore that goes for treatment; but there is not one mill in fifty at which this trouble is taken. Nor is it necessary. At most, the computation can only be approximate, because the condition of the ore will vary with almost every car-load, some being dry and some wet, some dense and pyritous and some porous and friable. Sufficient accuracy for all practical purposes is

attained by adopting cars of a standard size, filling them to the same average depth, and taking tally of the number delivered, after having computed the mean weight of their contents by weighing a number from different parts of the mine. To weigh every truck-load would occupy considerable time, and the results would not be commensurate with the cost entailed. Milling statistics are in consequence to be considered only as relative and not absolute.

**Sampling.**—The attempt is sometimes made to obtain regular samples of the milling dirt for assay, as a check upon the efficiency of the gold extraction processes employed. This may best be done by taking a shovelful periodically from the shoot of each bin, passing the whole through a small dry-crushing mill, in the proportion of about 1 cwt. from every 10 t. The bulk sample so obtained is quartered down until a representative assay sample can be taken. The results, however, can never be accurate, because a true sample taken while the ore is in such a coarse condition is impossible. Most mill managers satisfy themselves with estimating the milling value of the ore by adding the actual extraction to the contents of the tailings, the sum of these two being supposed to represent the original assay of the ore. This somewhat arbitrary method of calculation is probably quite as correct as sampling the milling dirt.

In custom mills, however, sampling is a matter of necessity, as the mill often buys the ore outright, paying a percentage (usually about 90 %) of the ascertained value. Custom milling, which is the salvation of many small fields, and might well be extended, has received its chief development in the United States, where many public sampling works now exist.

A simple automatic sampler, on the principle of the split shovel, consists of a narrow spout placed in the middle of the outlet from a bin, as shown in Fig. 57. When the handle *a* is pulled out, releasing a stream of ore so that it falls into truck *b*, the sheet-iron spout *c* deflects a portion of the stream on to the floor at *d*, whence it is collected and quartered down in the usual way.

Another easily-made device is a tube with a series of divisions in it. The diameter of the pipe must be adapted to the volume of ore passing through, so that it shall be reasonably filled. In large works, a common size is 8 in. diam., but one used by the author for sampling dried concentrates was only 2 in. diam. At suitable intervals in the tube, depending on its diameter, it is split by partitions which so alternate with each other that the first halves the stream, the next (at right

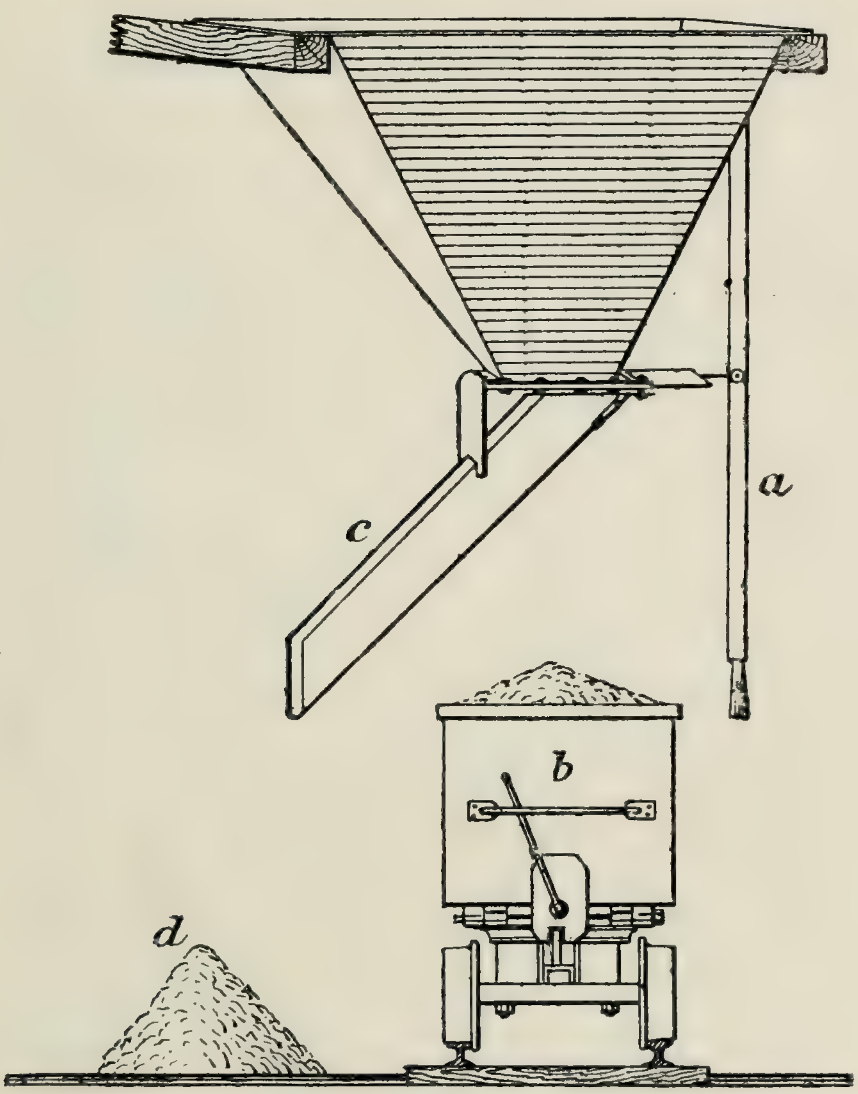



FIG. 57.—DRY SAMPLER.

angles) quarters it, and so on. In this way it is quite easy to abstract from the stream $\frac{1}{16}$ or $\frac{1}{32}$ of the whole bulk, and to pass that fraction, forming the sample, directly into a bucket, sack, or truck, according as the scale of operations may require. It can even be arranged to deliver two or more separate samples simultaneously. Like the apparatus shown in Fig. 57, it can be made by any whitesmith, or a more complicated and elaborate arrangement, as illustrated in Fig. 58, may be purchased from mining machinery firms.

The two machines indicated above are applicable solely to material in the condition of fine and dry powder. Coarse or damp material would block the passages and entirely vitiate results.

The Vezin sampler, Fig. 59, is a much more pretentious implement, and can be adapted for dealing with anything from the finest dust to 6 in. diam. It consists of two hollow truncated cones *a*, joined at their bases, and is supported and re-

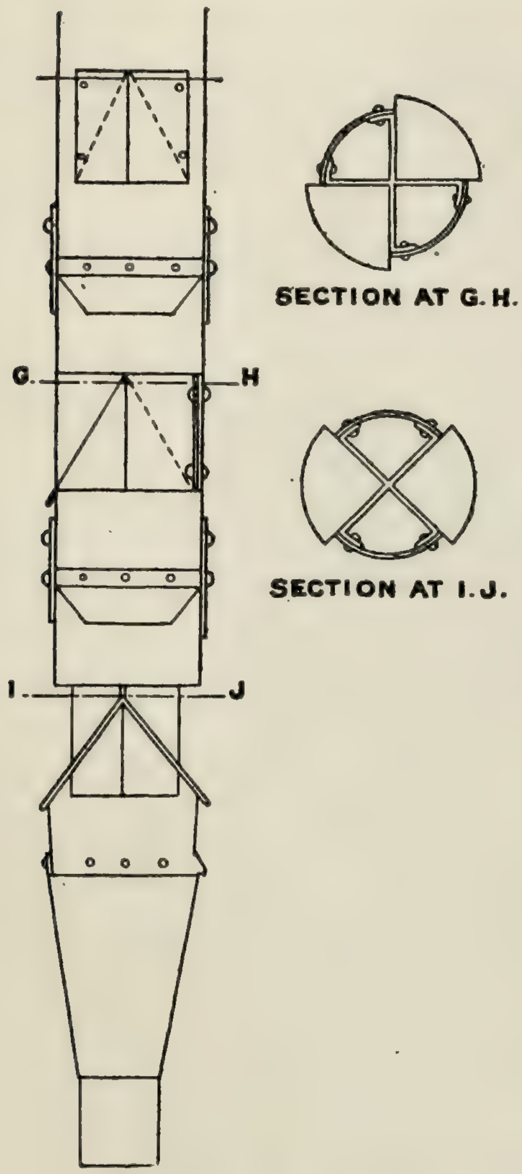



FIG. 58.—DRY SAMPLER.

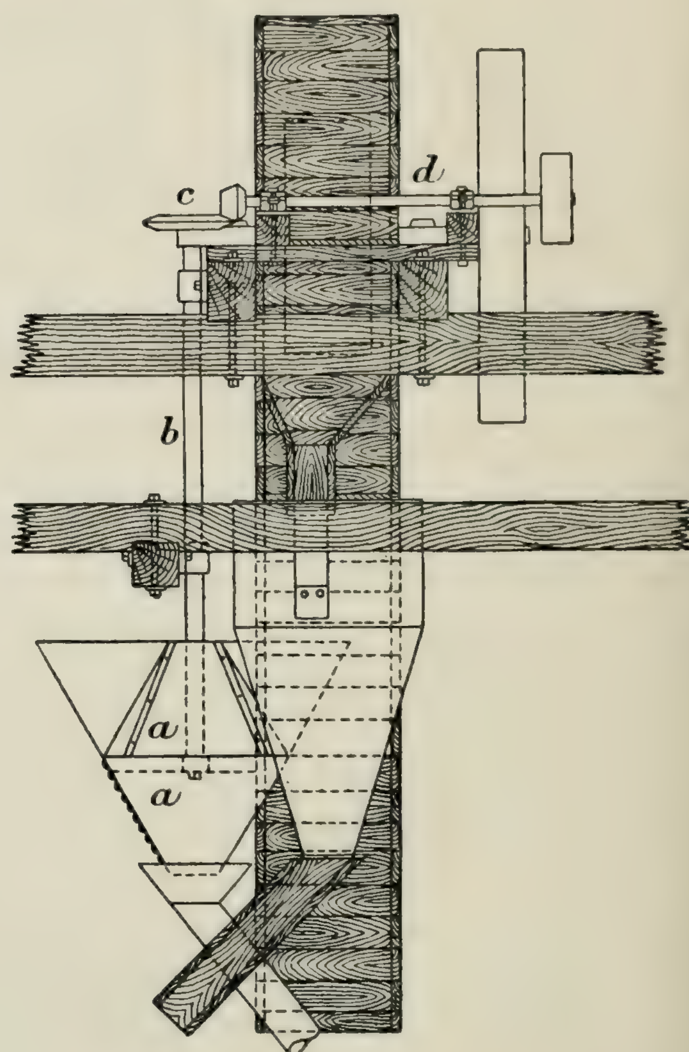



FIG. 59.—VEZIN DRY SAMPLER.

volved by a vertical spindle *b* driven by bevel gear *c* from a horizontal shaft *d*. The upper cone has one or more scoops, in the form of a sector, which, passing through a falling stream of ore, cuts out a sample, and delivers it into the interior of the cones, to be taken away by a spout to the sample bin or floor, while the rejected portion falls into a hopper, and is spouted to storage bins or cars. It moves in one direction with uniform speed, and can be run much faster than a sampling machine with a reciprocating motion ; and there is

no danger from breakage due to blows or jars. The sample taken is rhomboidal in section, not wedge-shaped; and if larger samples are required, a larger portion is not taken the same number of times, but the same portion is taken more frequently. The machine is well adapted for taking duplicate samples for the purpose of checking them against each other, and requires but a very short drop of the ore. With a short inclined spout to compact and equalise the discharge from an elevator or a feeding shoe, or a revolving cylinder to equalise the discharge from a slow motion elevator, the drop of the ore need not be more than 1 in. In the single form, the machine weighs about 600 lb. and costs 30*l.*; in the duplicating form, 1100 lb., and 40*l.*

The sampling of tailings is a matter that should never be neglected, because it is the only real check upon losses incurred in treatment. Hand-sampling is the most common procedure, a cupful of tailings being taken periodically from each issue, or a larger quantity from the general tailings launder. This practice has two great drawbacks—it relies upon human memory and personal precision, and makes the operative a tell-tale on his own failings. Hence, half its value is destroyed. If taken at long intervals, it is no sample; if at short intervals, it demands so much attention that it is certain to be neglected. Carelessness in taking the sample, by allowing the vessel to overflow and thus cause concentration, will vitiate it entirely. Inattention to duties during the night shift, for example, should be revealed in the tailings assay, but if the delinquent is to take his own sample he may be relied on to screen himself if possible. Therefore, the only true means of sampling tailings is an automatic one, out of reach of the mill hands altogether.

There are quite a variety of ways by which automatic sampling of mill pulp may be attained, and that too with a regularity and completeness leaving nothing to be desired, and with a frequency governed by the needs of the ore and the volume of sample which can be conveniently handled.

The author has adopted with success for some years the arrangement shown in Fig. 60, which he found in use in a Californian mill, where it was giving entire satisfaction. The

sampling mechanism proper is a long narrow trough of sheet iron *a*, measuring about 1 in. wide × 3 in. deep (to avoid splash), and long enough to reach completely through the stream of pulp flowing in the launder *b*. The sampler *a* is riveted to an iron rod *c*, which is pivoted at *d*, and whose short end *e* is periodically encountered by a hard steel pin *f* on the periphery (or better still in a slot on one of the spokes as at *g*, which admits of adjustment when the pin becomes worn) of the toothed wheel *h*. This last is rotated by the worm *i* through pulley *k*. Any source will do from which

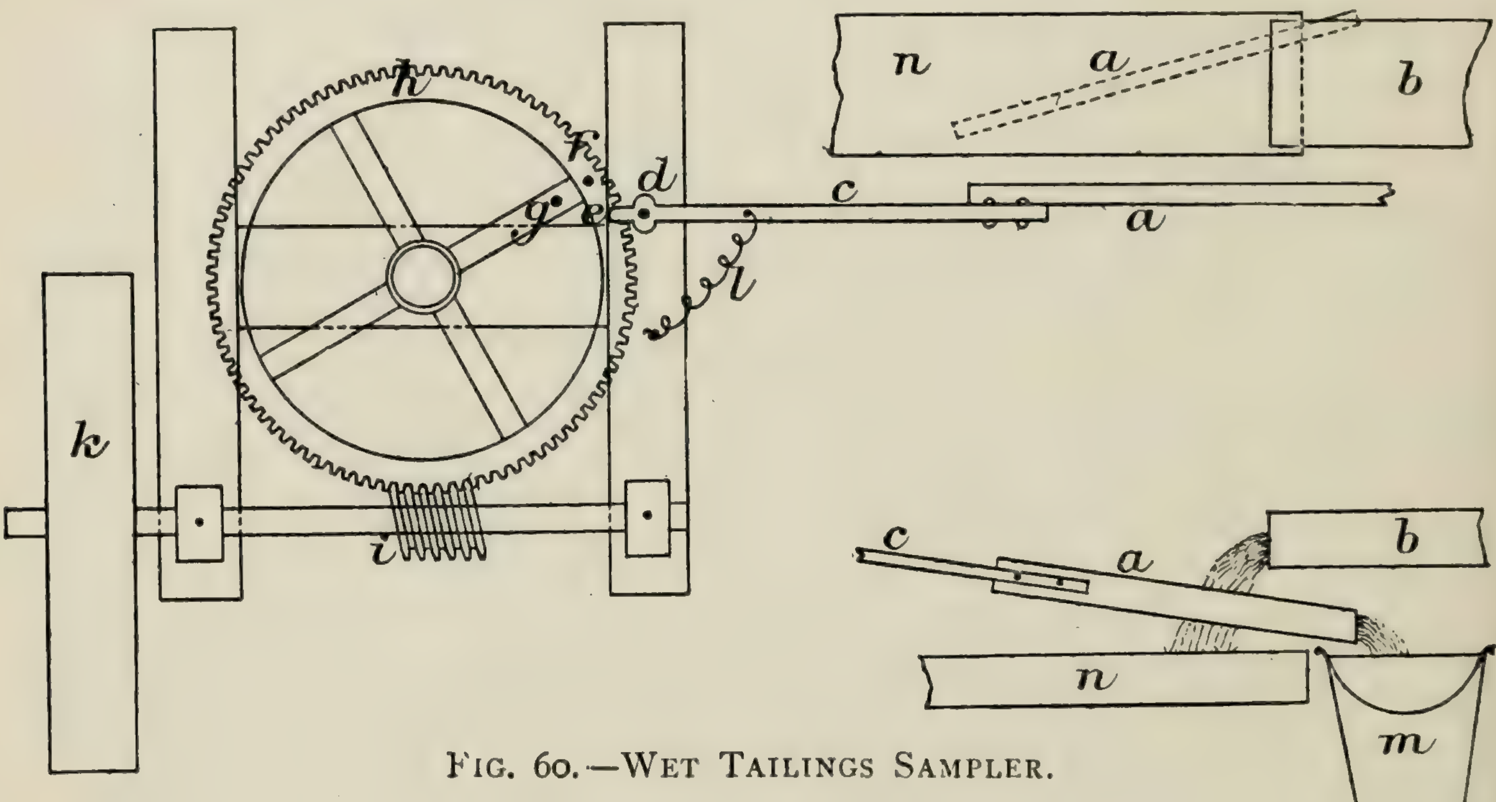



FIG. 60.—WET TAILINGS SAMPLER.

to take motive power, but it may best be had from some shaft which has periods of movement and rest exactly according with those of the battery itself, such as a vanner countershaft, or a tailings-wheel shaft, or the line shaft of the battery, bearing in mind that the speed of *h* needs to be very slow, every revolution giving a sample. A spring at *l* serves to bring the sampler back to its place when the end *e* is released by the pin *f* or *g*. The sampler *a* passes across the entire stream, and cuts out a portion from the whole width of the launder, transferring it to the bucket *m*, which is removed with its total contents for drying and splitting down in the assay room. When the sampler is not operating, the pulp flows on without interruption into the launder *n*. The whole

arrangement should be enclosed in a large cupboard to which the mill men have no access. The buckets *m* (one in reserve) should bear a special mark, and never be used for any other

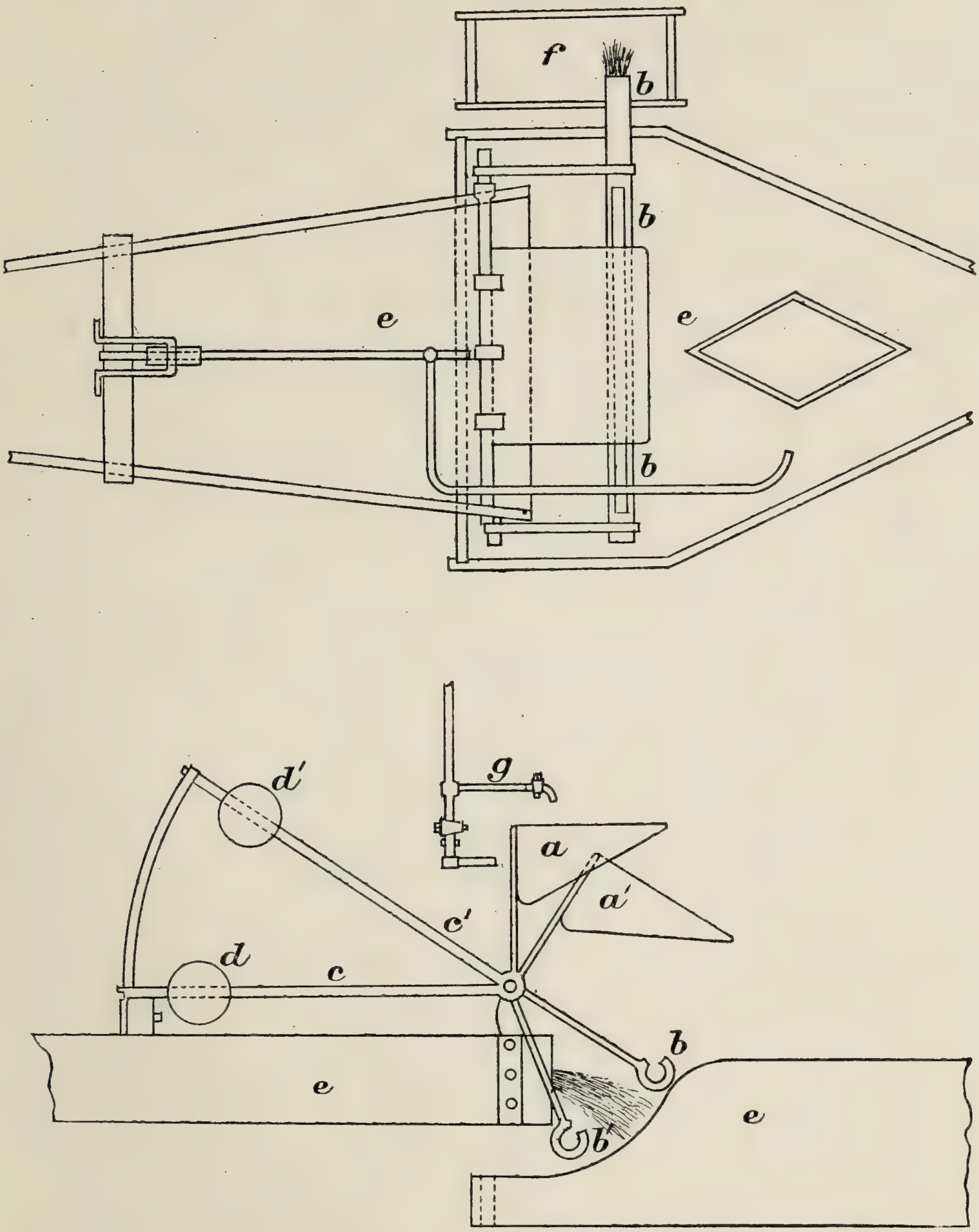



FIG. 61.—WET TAILINGS SAMPLER.

purpose; they are handled only by the assay-office staff. When properly arranged, the contrivance makes no slop, and the sample taken by it is a true one, there being neither concentration by splash or swirl, nor contamination by care-

lessness. A very slight drop is necessary between issue from launder *b* and intake by launder *n*,—just enough to give clearance for the sampler to pass laterally between.

Instead of a positive mechanism, reliance may be placed on a modification of the Barker mill for securing periodicity, but this is always open to the objection of likelihood of interference with the regularity of sampling by the numerous causes which may influence the flow of a tiny stream, such as frost, rust, suspended matter, and so on. Nevertheless the apparatus shown in Fig. 61 is much favoured in the Transvaal. The sampler in this case is a slotted pipe $b\,b^1$ carried on the short end of a pivoted bar $c\,c^1$, and tilted into the stream of pulp flowing from one section to another of the launder *e* by the scoop $a\,a^1$. The long arm supports a counterpoise $d\,d^1$, which brings *a* back under the water-tap *g* for refilling. The sampler discharges into a box *f*. Notwithstanding that the launder *e* is much widened, in order to render the stream of pulp more shallow, and thus facilitate its entrance into the sampler, it is a very open question whether nearly as true a sample is got in this way as by the means previously described. Practically the same amount of drop is necessary to bring the sampler into play.

**Mill Labour.**—First-class mill-hands, or amalgamators, as they are often called, are nowhere abundant. They need to be mechanics, robust, of sober habits, and of unimpeachable honesty; consequently they command and are entitled to high wages. In America, their wages range from 15*s.* to 20*s.* per 12-hour shift; in Australia, from 7*s.* 6*d.* to 9*s.* per 8-hour shift; and in the Transvaal, they average 17*s.* 4*d.* per 12-hour shift. On the opposite page are a few examples of the distribution of labour in representative mills.

**Milling Costs.**—Any attempt at a strict comparison between milling costs at various centres must fail, because no account can be taken of the nature of the ore, which, after all, is the prime factor. Nevertheless, while making allowance for this obvious omission, it is thought that the table of milling

MILL LABOUR PER DIEM.

| — | Stamps. | Breaker. | | Battery. | | Concentrators. | | Mechanics. | | Labourers. | | Total. | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | No. | Total Wages. | No. | Total Wages. | No. | Total Wages. | No. | Total Wages. | No. | Total Wages. | No. | Wages. |
| California: | | | s. d. | | s. d. | | s. d. | | s. d. | | s. d. | | s. d. |
| North Star . . . | 40 | 1 | 12 6 | 2 | 37 6 | 2 | 25 0 | 1 | 9 4 | .. | .. | 6 | 84 4 |
| Dakota: | | | | | | | | | | | | | |
| Homestake . . | 100 | 2 | 25 0 | 5 | 75 0 | .. | .. | 6 | 86 6 | 6 | 70 10 | 19 | 257 4 |
| Do. . . | 120 | 2 | 25 0 | 5 | 75 0 | .. | .. | 6 | 90 6 | 9 | 106 3 | 22 | 296 9 |
| Do. . . | 160 | 2 | 25 0 | 5 | 75 0 | .. | .. | 6½ | 104 2 | 10 | 116 8 | 23½ | 320 10 |
| Do. . . | 160 | 3 | 31 3 | 6 | 79 2 | .. | .. | 5½ | 75 0 | 6 | 60 5 | 20½ | 245 10 |
| New South Wales: | | | | | | | | | | | | | |
| Lucknow . . . | 30 | 1 | 7 6 | 3 | 27 0 | 3 | 27 0 | .. | .. | 2 | 13 0 | 9 | 74 6 |
| New Zealand: | | | | | | | | | | | | | |
| Saxon . . . . . | 33 | .. | .. | 3 | 24 0 | 3 | 10 0 | .. | .. | 6 | 35 0 | 12 | 69 0 |
| Transvaal: | | | | | | | | | | | | | |
| New Primrose . . | 160 | .. | .. | 12 | .. | .. | .. | 6 | .. | 31 | .. | 49 | 500 0 |
| Simmer and Jack . | 100 | .. | .. | 9 | .. | .. | .. | 3 | .. | 4 | .. | 16 | 268 0 |

costs presented below cannot fail to be of interest, if only as showing how widely the figures vary.

MILLING COST PER TON.

| — | Stps. | Duty. | Labour. | Power. | Stores. | Maint. | Water. | Total. |
|---|---|---|---|---|---|---|---|---|
| Alaska: | | t. | s. d. | s. d. | s. d. | s. d. | s. d. | s. d. |
| Mexican .. .. .. | 60 | 3·68 | 7½ | 4 | 4½ | .. | .. | 1 4 * |
| Treadwell .. .. | 240 | 2·90 | 7¾ | 4 | 10¾ | .. | .. | 1 11 * |
| Brazil: Faria .. .. | .. | .. | .. | .. | .. | .. | .. | 2 9½ * |
| California: | | | | | | | | |
| Gover .. .. .. | 20 | .. | 8 | 10 | 6 | .. | .. | 2 0 |
| Keystone .. .. .. | .. | .. | 1 1½ | .. | 7½ | 7 | 9½ | 3 1½ |
| North Star .. .. | 40 | 1·59 | 8 | 1 1 | 5 | .. | .. | 2 2 |
| Plymouth .. .. | .. | .. | .. | .. | .. | .. | .. | 1 7½ |
| Utica .. .. .. | 60 | .. | .. | .. | .. | .. | .. | 7 † |
| Colorado: | | | | | | | | |
| Hidden Treasure .. | 75 | .. | .. | .. | .. | .. | .. | 3 3 * |
| Dakota: | | | | | | | | |
| Homestake .. .. | 80 to 160 | .. | 10½ to 1 0½ | 1 1½ | 1½ to 3½ | 4½ to 6 | 8½ | 3 2½ to 3 7½ |
| India: | | | | | | | | |
| Champion .. .. | 140 | .. | .. | .. | .. | .. | .. | 5 11½ |
| Coromandel .. .. | 20 | .. | .. | .. | .. | .. | .. | 5 10½ |
| Mysore .. .. .. | 120 | 2·35 | 1 8½ | 2 4 | 11¼ | .. | .. | 4 11¾ |
| Nundydroog .. .. | 70 | 2·68 | 1 3 | 2 3 | 10½ | .. | .. | 4 4½ |
| Mexico: | | | | | | | | |
| Mesquital .. .. | 50 | .. | 2 0 | 1 5¾ | 1 4 | .. | .. | 4 9¾ |
| New South Wales: | | | | | | | | |
| Gibraltar .. .. .. | .. | .. | .. | .. | .. | .. | .. | 4 6 * |
| Lucknow .. .. | 30 | 1·37 | 6 | 6½ | 1 | 8½ | .. | 1 10 |
| Myalls .. .. .. | 40 | 2·59 | 11½ | 3½ | 4½ | .. | .. | 1 7½ |
| New Zealand: | | | | | | | | |
| Saxon .. .. .. | 33 | 2·12 | 11½ | 3½ | .. | .. | .. | 3 7¾ |
| Nova Scotia: | | | | | | | | |
| Brookfield .. .. | 20 | 2·00 | 1 6¾ | 7½ | 1¾ | 3½ | .. | 2 7½ |
| Richardson .. .. | 40 | 2·45 | 10¾ | 4 | 2 | 1 | .. | 1 5¾ |
| Queensland: | | | | | | | | |
| Brilliant Block .. | 40 | 1·56 | 2 9½ | 1 6½ | 10¼ | 11 | ... | 6 1¼ |
| Day Dawn P.C. .. | 50 | .. | 7 3 | 3 5½ | 2 | 1 7 | .. | 12 5½ |

* Includes concentration. † Exclusive of power.

MILLING COST PER TON—*continued.*

| — | Stps. | Duty. | Labour. | Power. | Stores. | Maint. | Water. | Total. |
| --- | --- | --- | --- | --- | --- | --- | --- | --- |
| Transvaal: | | t. | s. d. | s. d. | s. d. | s. d. | s. d. | s. d. |
| Angelo .. .. .. | 100 | 4·67 | .. | .. | .. | .. | . | 2 8 |
| Bonanza .. .. .. | 40 | 4·89 | .. | 1 4¾ | .. | 2¼ | .. | 4 7 |
| Crown .. .. .. | 120 | 4·60 | 11¼ | 8 | 10¾ | 5¼ | .. | 2 11¼ |
| Crown Deep .. .. | 160 | 4·97 | .. | .. | .. | .. | .. | 3 7 |
| Driefontein .. .. | 110 | 4·70 | .. | .. | .. | .. | .. | 2 8 |
| Durban Roodepoort | 80 | 4·51 | .. | .. | .. | .. | .. | 3 2½ |
| Do. Deep .. .. | 50 | 4·90 | .. | .. | .. | .. | .. | 4 9½ |
| Ferreira .. .. .. | 80 | 4·73 | .. | .. | .. | .. | .. | 2 8 |
| French Rand .. .. | 60 | .. | 1 1½ | 7½ | 8½ | .. | .. | 2 5½ |
| Geldenhuis .. .. | 120 | 5·02 | 10½ | 5 | 5 | .. | 6 | 2 2½ |
| Do. Deep .. | 190 | 4·52 | .. | .. | .. | .. | .. | 2 11 |
| Glen Deep .. .. | 100 | 5·16 | .. | .. | .. | .. | .. | 3 10½ |
| Glynn's Lydenburg | 20 | 4·19 | .. | .. | .. | .. | .. | 2 0 |
| Johannesburg Pioneer | .. | .. | 1 10 | 1 9½ | 1 0 | 9½ | .. | 5 5 |
| Jumpers .. .. .. | 100 | 4·27 | 1 8½ | 1 0 | 6½ | 1 3 | .. | 4 6 |
| Do. Deep .. | 85 | 4·92 | .. | .. | .. | .. | .. | 4 4½ |
| Langlaagte .. .. | 200 | 4·48 | 7½ | 7¼ | 4¾ | 10 | .. | 2 5½ |
| Do. Deep .. | 100 | 5·54 | .. | .. | .. | .. | .. | 4 4 |
| New Comet .. .. | 90 | 4·63 | .. | .. | .. | .. | .. | 2 4½ |
| New Goch .. .. | 60 | .. | .. | .. | .. | .. | .. | 4 9 |
| Nourse Deep .. .. | 100 | 4·81 | .. | .. | .. | .. | .. | 3 5½ |
| Paarl .. .. .. | 60 | 4·19 | .. | .. | .. | .. | .. | 2 9½ |
| Robinson .. .. | 140 | 4·40 | 1 2¼ | 8¾ | 6¾ | .. | .. | 2 5¾ |
| Rose Deep .. .. | 150 | 4·90 | .. | .. | .. | .. | .. | 2 9½ |
| Sheba .. .. .. | 200 | 2·85 | 2 4½ | .. | 1 8¾ | 1 7½ | .. | 5 8¾ |
| Transvaal Estates .. | 60 | 3·00 | .. | .. | .. | .. | .. | 1 8 |
| Van Ryn .. .. .. | 160 | .. | .. | .. | .. | .. | .. | 3 0 |
| Village .. .. .. | 100 | .. | .. | .. | .. | .. | .. | 2 8¾ |
| Windsor .. .. .. | 50 | .. | .. | .. | .. | .. | .. | 4 0 |
| Witwatersrand .. | .. | .. | 11 | 5¼ | 9¾ | .. | .. | 2 2 |
| Venezuela: El Callao | 60 | 2·91 | 1 5 | 1 3¼ | 8 | 1 0¾ | .. | 4 5 |
| Victoria: | | | | | | | | |
| Clunes .. .. .. | 60 | 2·80 | .. | .. | .. | .. | .. | 2 0 |
| Magdala .. .. .. | 30 | 2·24 | 3 1½ | 8¼ | 7 | 11½ | ¼ | 5 4½ |
| Western Australia: | | | | | | | | |
| North Boulder .. | .. | .. | 3 3¾ | 1 8 | 7¼ | 3¼ | 1 9¼ | 7 7½ |
| White Feather .. | 20 | .. | .. | .. | .. | .. | .. | 6 5 |

# CHAPTER IV.

## WET MILLING—OTHER METHODS.

WHILE no machine yet introduced has been successful in displacing the stamp-battery as a wet crusher (with or without simultaneous amalgamation), the inventive faculties of engineers have produced many types of wet-crushing apparatus, which find a sphere of usefulness here and there under special circumstances. While for a large establishment nothing can compete with gravitation stamps, it must be admitted that for undertakings of very limited dimensions, and where the work is more or less tentative, a machine demanding less outlay for foundations, and of a more portable type, has claims to attention, notwithstanding its inferior capacity. Now and again also, the ore is so soft, or carries so much clay, that it is not too well adapted for the action of stamps, and then the less vigorous machine has an opening.

A rough classification of these inventions, without any pretence of embracing them all, for their name is legion, is as follows :—

(*a*) Those operating by heavy rolling weights, such as the Chilian mill and its imitators (the Wiswell pan, Bryan mill, Neate's dynamic grinder, etc.).

(*b*) Those operating by a multiplication of cylindrical or spherical crushing faces, as the Huntington mill and the ball mill.

(*c*) Those operating by stamps at great speed, resembling a steam hammer, and known as steam stamps.

Descriptions will be confined to those which have been found to warrant some practical adoption.

**Chilian Mills.**—Although the Chilian mill is one of the primitive machines used in Mexico and South American

countries from very early times, its modern form bears but small resemblance to the crude prototype. Properly run, it is a useful appliance under some conditions. It is urged against it that the weight and bulkiness of the runners hinder transportation, that the wave created by the runner carries the material to be crushed away from its action, that sliming is encouraged, and that the friction at the end collars of the

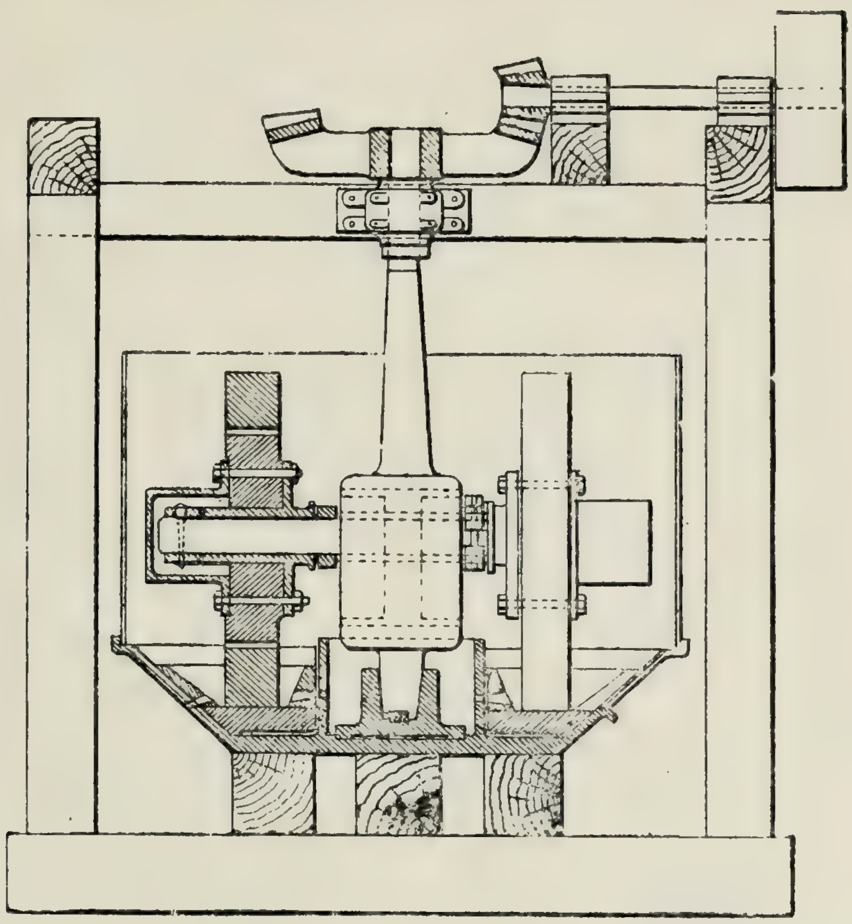


FIG. 62.—CHILIAN MILL.

spindles involves much wear of metal and waste of power. However these indictments are only partially true, and the machine as made by W. P. Blake, New Haven, Conn., is successfully applied to crushing very hard stone from $\frac{1}{4}$ in. gauge till it will pass a 40-mesh screen. Each runner of the mill, as shown in cross section in Fig. 62, weighs about 1 t. and measures 4 ft. diam. and 8 in. across the face, the distance between outsides being 4 ft. 2 in. The central spindle carrying the horizontal axis, on which the runners revolve, makes 40 rev. a minute. The tyres of the runners are of hard white iron 8 in. thick, and the bed of the pan carries segmental dies of best chilled iron. The pan is surmounted and enclosed

by sheet iron 4 ft. high, to retain the splash of pulp. The latter is discharged from the pan by a current of water on to the inner periphery of revolving screens of 40 mesh, quite independent of the mill. The rate of output measured on some thousands of tons is $3\frac{3}{4}$ t. of extremely hard and tough quartzite per hour per mill, at a low cost for power. The wear of iron from crushing faces amounts to ·12 lb. per ton of ore milled, which is appreciably less than with stamp batteries. Chrome and manganese steel would seem to be eminently adapted for the pan bed and runner tyres.

Very much larger examples of the Chilian mill are used on the Ural goldfields, the pans reaching 15 ft. diam. and 30 in. deep, and the runners (3 in number) measuring 4 ft. diam. and 12 in. wide and weighing 3 to 4 t. Steel hoops are placed on the runners, and the beds are of steel segments. The discharge takes place through screens in the outer periphery of the pan, and at 12 in. above the bed. The speed is only 12 to 18 rev. a minute. With free milling ores, mercury is added to the pan.

An "improved" Chilian mill known as the Reynolds, made by the E. P. Allis Co., Milwaukee, is shown in Fig. 63. Its main feature is the self-adjustment of its rollers. The roller *a* is mounted with its trunnions in the end bearings of the fork-shaped steel casting *b*. The shank *c* of this casting is turned true, and moves in the bearings. The bearings *d* of the fork are set a little behind the centre of the shank *c*. The principle of this design is, that as the speed of the rolls increases, so also does the centrifugal force, in direct proportion to one another. Thus it will be seen that on account of the position of the shank *c* in relation to the roll axis, the centrifugal force exerts a tendency on the roll and its axis to assume a position that is not radial with the centre shaft. This position causes a tendency for the roll to run to the centre of the mill in direct opposition to the centrifugal force. The result is the roll keeps itself in perfect equilibrium between the outward tendency due to centrifugal force, and the inward crowding pressure caused by the position of the roll axis combined with the traction power of the roll. A lug *e*, which moves between

two ribs of the upper casting *f*, prevents any excessive side motion which might happen by accident. The casting *f*, which is keyed to the upright shaft, is the driver of these rolls. The rolls are sometimes made hollow, so that they can be

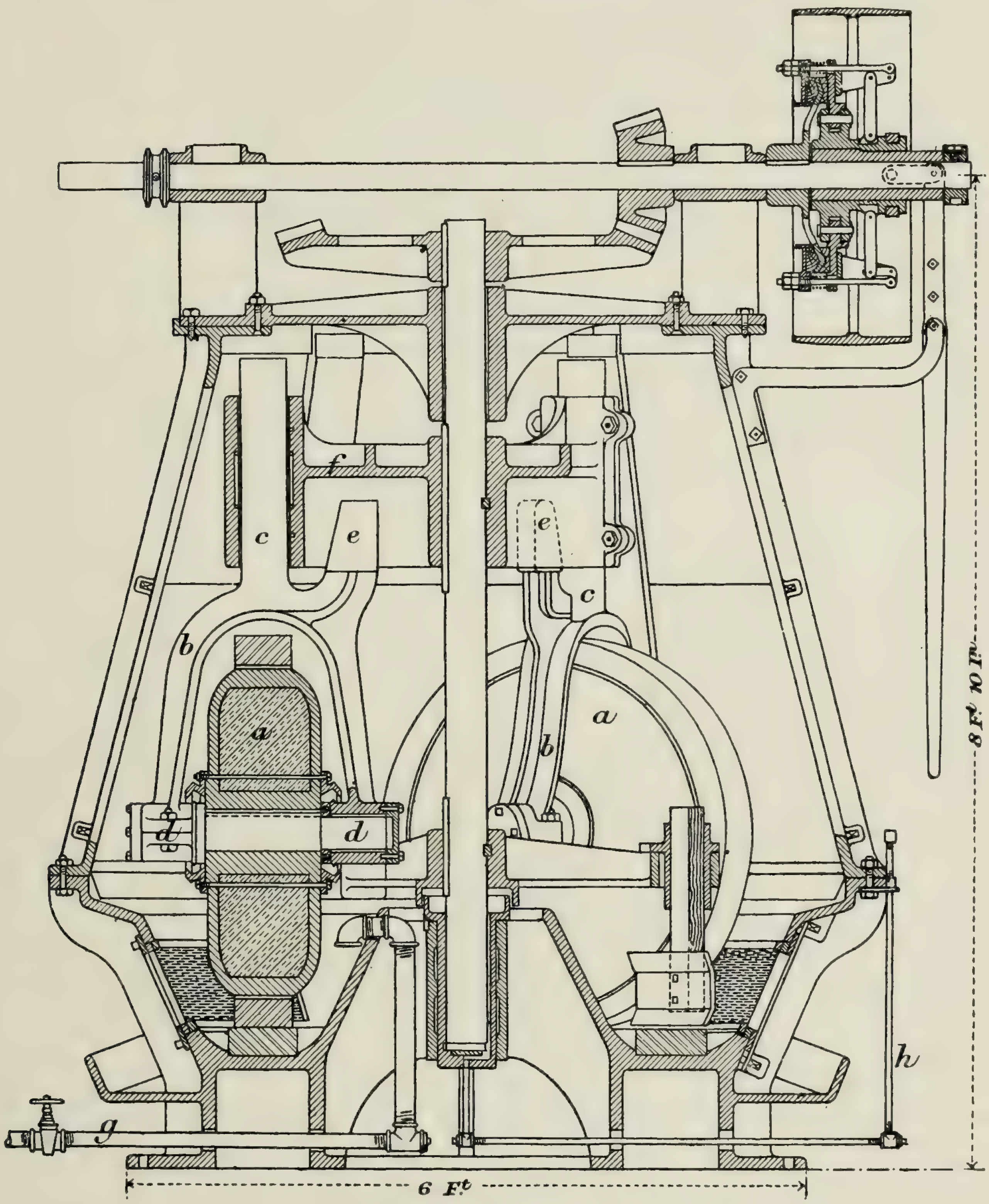



FIG. 63.—REYNOLDS CHILIAN MILL.

filled with lead, to increase their effective weight. Water is supplied by pipe *g* to the bed of the mill ; and lubricant is fed to the footstep bearing by pipe *h*.

The same makers turn out the "reliance" Chilian mill, illustrated in Fig. 64. Its motive power is applied from

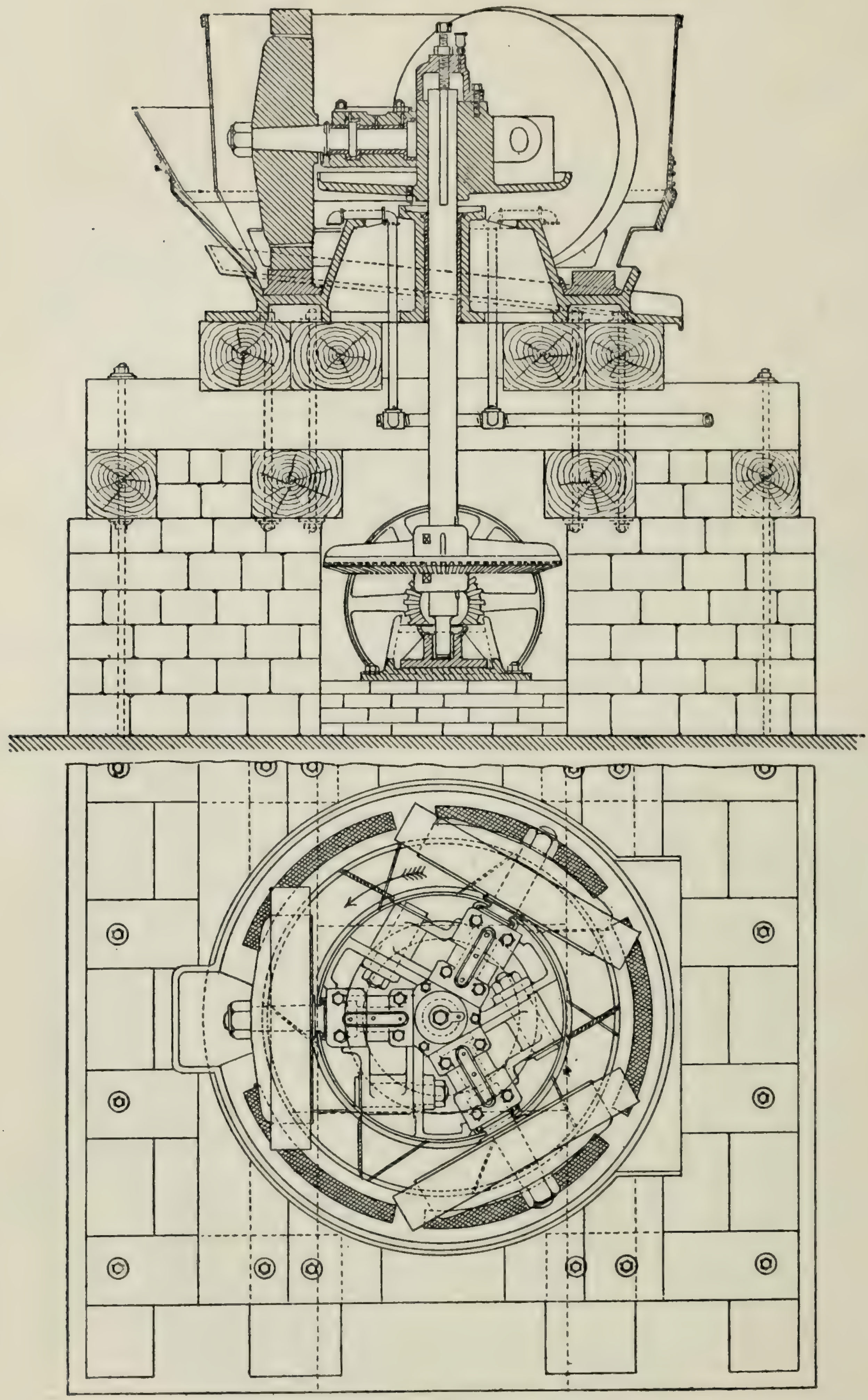


FIG. 64.—RELIANCE CHILIAN MILL.

below, consisting of counter-shaft, pulley and bevel gearing. The pulley can be driven from a line-shaft located above the mill, and as this shaft may be run at a fair speed, its weight and cost can be greatly reduced. Where a number of mills are driven from one line-shaft, a clutch-pulley may be provided to drive each mill, so that any one may be stopped and started without disturbing the others. Another feature is that the means employed for attaching the rollers to the vertical driving-shaft consist of an axle-bearing so designed as to permit the rollers to swing in a vertical plane, thereby relieving the upright shaft of a side strain, which is always present when rollers are rigidly attached to the upright shaft. This method also permits each of the 3 rollers to adjust itself to the level of the ore bed, taking the load from the vertical shaft and placing it on the ore. The tyres and dies are made of special steel ; the bearings are lined with anti-friction composition, and are provided with automatic lubricators with generous drip-pans, to keep drippings out of the mortar. The periphery of the pan is cast with openings, to which screens are applied, and provision is made for a set of water-pipes. When for milling free gold ores, copper table and mercury trap can be attached. The 5-ft. mill has rollers approximating in weight 5000 lb. each ; its set of tyres and dies will crush 6000 to 8000 t. of average ore, lasting about a year in constant use. The average capacity of this size mill is 25 to 30 t. per day, and it requires 6 h.p.

A modified Chilian mill, known as the Wiswell pan, is much in favour for crushing cemented gravel (similar to the banket of the Rand) in the Black Hills of Dakota. Essentially, it consists of a pan with an annular groove in the bed, of conical section, in which revolve four heavy (2000 lb. each) iron wheels, with faces shaped to correspond with the groove in which they travel, as shown in Fig. 65. The ore, roughly hand-broken, is fed by the hopper *a* through a slot in the sheet-iron casing *b* which surrounds the pan, and finds its way into the annular groove *c* of the bed, where it is pulverised by the grinding action of the wheels or runners *d*, water being admitted in the usual manner. A small amount of mercury

is fed into the pan at intervals, and gathers at the bottom of the groove *c*, whence it can be drawn off as thin amalgam by the tap *e*. About one-third of the peripheral sheet-iron casing *b* is replaced by slotted iron screens, through which the pulp is discharged. The iron bed is made in three sections, which bolt together securely; and the groove *c* is furnished with a false bottom, in pieces of convenient size for handling. Copper-plate tables are provided below the screen discharge, and catch some additional gold, but the major portion is arrested by the mercury in *c*. A small projection of the framework which actuates the runners *d* serves to regulate the feed by agitating the hopper. The machine is altogether remarkably well adapted for the somewhat limited sphere of application open to it. It is very simple and inexpensive, has nothing to get out of repair, requires but little power and


FIG. 65.—WISWELL PAN.

attention, and exercises a certain amount of grinding as well as a crushing effect, which is in some instances very beneficial. The wear and tear or consumption of iron is very small as compared with stamps; and the shape of the runners reduces the tendency of the pulp to flow in front of them, as is more or less the case with Chilian mills. But it cannot be employed on an ore which is in any degree pyritiferous, as the heavier mineral will not issue at the periphery, but accumulates in the groove of the bed.

**Huntington Mill.**—In the Huntington centrifugal roller mill, shown in Fig. 66, the vertical spindle *a*, driven from beneath by bevel gearing *b*, supports a circular horizontal cast-iron frame or "disc-driver" *c*, from which are suspended the crushing-roller *d* and the scrapers, by means of journals forming bearings for the ends of yokes. The rollers *d* are free to swing in a radial direction with reference to the vertical

spindle *a*, and, when in action, centrifugal force carries them hard against the steel ring-die *e*, fixed to the inside of the rim *f*. Each roller consists of a steel ring, which rotates on its own axis. A clearance of about 1 in. is left between the lower edge of the rollers and the bed of the pan, so that the former pass freely over the layer of mercury there provided. Scrapers

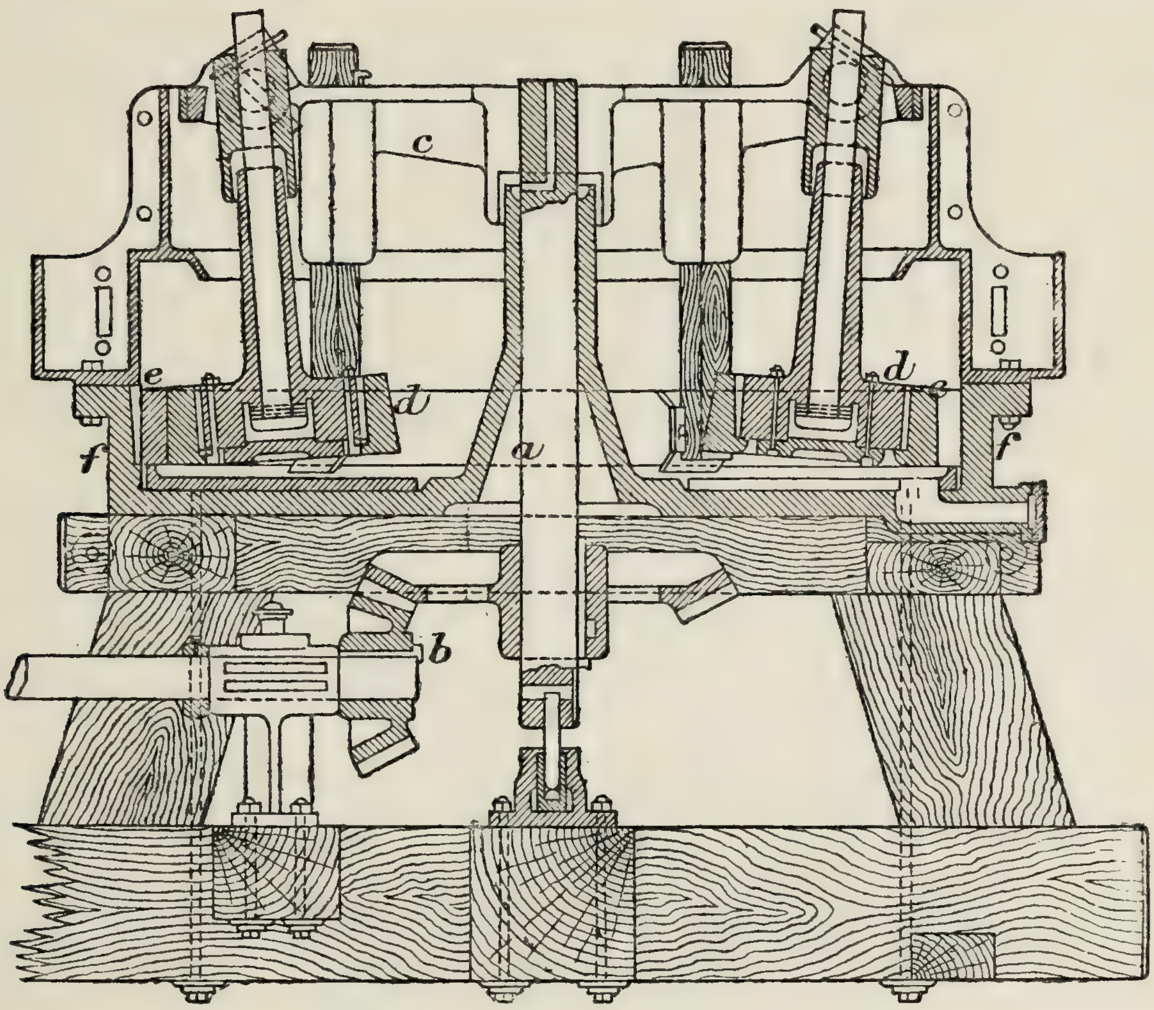



FIG. 66.—HUNTINGTON MILL.

set at varying distances from the centre help to drive the ore against the steel ring-die *e*. The ore is fed in with water through a hopper, and is crushed between the fixed ring-die *e* and the rotating rollers *d*. Some 60 lb. or more of mercury is charged into the pan, and helps to arrest any free gold liberated, while the pulp issues through screens occupying about one half the total circumference of the mill.

As may be seen, the foundation necessary for the mill is very simple, and its erection is correspondingly facilitated; but nothing larger than ½ in. mesh should be fed to the machine, and it is quite unfit to deal with hard rock. For soft clayey ores, however, it is very well suited, where a stamp battery could not do itself justice.

**Ball Mill.**—The ball mill, so well known as a fine dry-crushing machine, has recently by an ingenious device, been adapted for wet milling. This form is made by the Krupp Grusonwerk at Magdeburg-Buckau, where the dry mill also originated. The wet mill is made in eight different sizes, as the dry mill is, each possessing about 30 % greater efficiency than the corresponding machine operating dry.

As may be seen from Fig. 67, the drum of the mill and its charge of steel balls are virtually the same in both cases, the main difference in the wet mill being in the casing *a*, which, at its lower part, is constructed like a pointed box or spitz-kasten. Clean water is introduced through the pipe *b*, which is connected with the sprays *c*, the latter being regulated by valves. These sprays throw water against the fine outer sieves of the drum, which are thus kept perfectly clear. Fresh water can also be introduced into the pointed box through the pipe *d* (when the mill is first started, etc.) ; and it may likewise be admitted in an upward stream to separate the fine slimes from the bulk of the ground product. The water level in the box is regulated by a slide *e*, which is adjusted by means of a toothed wheel *f* and hand wheel, in such a way that the drum of the mill is under water to the lowest level of the inner sieves. The object sought is to enable the scoops *g* to catch sufficient water to wash back the coarse ground material which rests on the inner and outer sieves, and which is thus returned to the inner part of the drum through channels provided for this purpose, as the drum rotates. If the water level is higher than the inner sieves, the scoops receive so much water that it overflows through the feeding nave which admits the material to be ground. The overflow *h* serves to regulate the level of the water in the mill. As only a very small quantity of water is to overflow here, the sprays *c* must only pass as much water as is necessary to cause the ground product to flow through the pipe *i* and to keep the water level in the box up to the height indicated. The water which overflows through *h* contains the slimes, and is usually conveyed away together with the coarser product through the pipe *i*. The slimes can, however, be separated

from the coarsely ground material by means of an upward flow of water through the pipe *d*. In this case, the finest

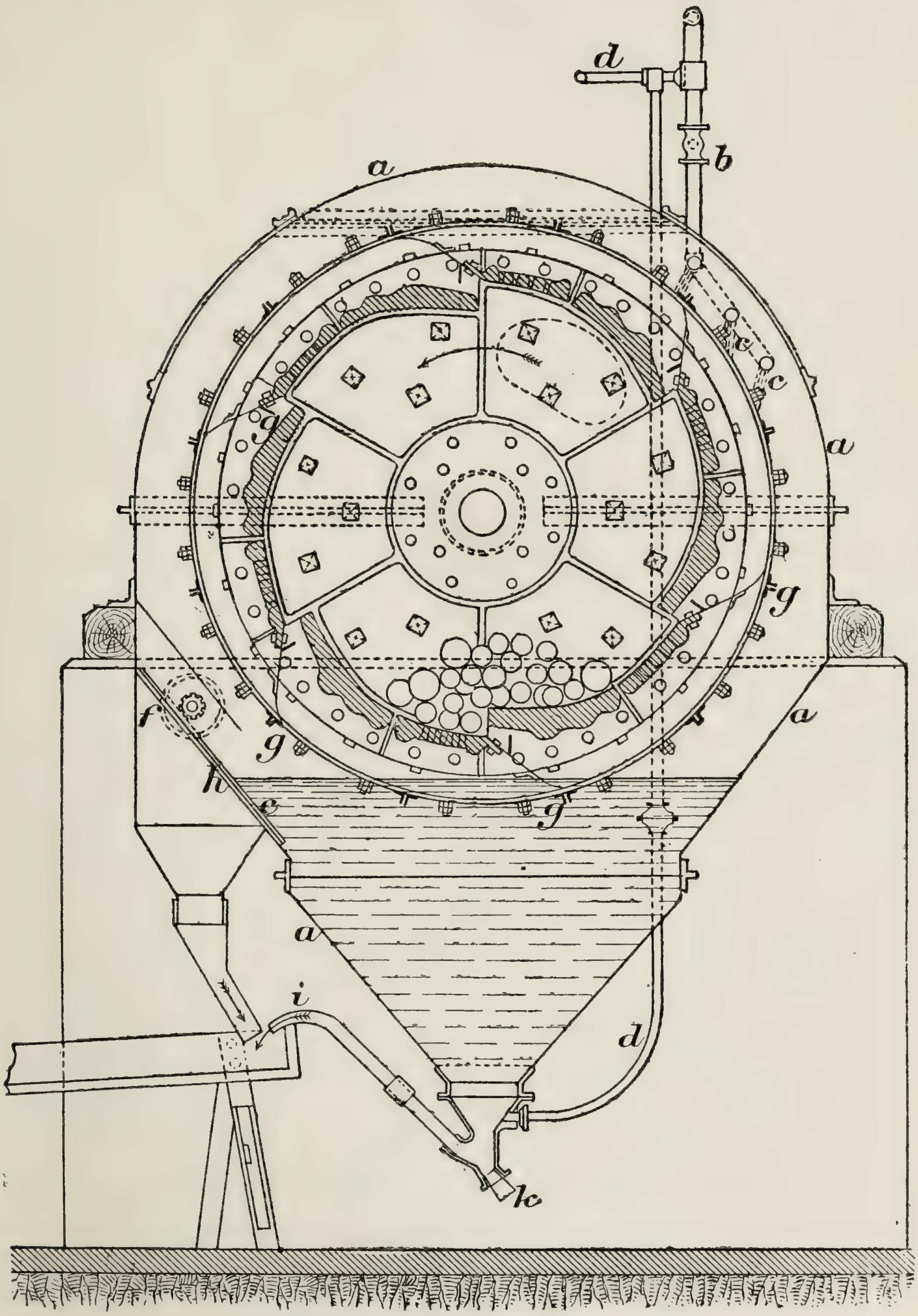



FIG. 67.—WET BALL MILL.

particles are, owing to the upward stream of water, carried away in the overflow *h* and kept separate from the less finely ground material. The pipe *i* must not be too high, or it

will tend to become blocked. Moreover, the breadth of the opening must be so chosen that the stream of ground material may be of suitable consistency. If this pipe be too narrow, it will easily become stopped up. If it be too broad, the discharge stream will contain too much water. On this account, every mill is provided with discharge pipes of different widths.

Should the pointed box become stopped up with sand, this can easily be cleaned out after removal of the stopper *k*. Attention must be paid to the direction in which the mill rotates; and the arrangement of the fresh-water connection *b* with the sprays *c* on the one side, and the slide *e* with the overflow *h* on the other side, must be designed strictly according to the drawing.

**Steam Stamps.**—Steam stamps of enormous capacity have long been used on the Lake Superior copper mines, for crushing to a much coarser mesh than is desirable in gold milling, and one was tried some years ago at the Homestake mills, Dakota. Its size was 26 in. by 11 in., and it was run at a speed of 95 22-in. strokes per minute, with a water-supply of about 900 gal. per ton. At 85 lb. steam pressure, it crushed 125 to 135 t. per 24 hours to pass No. 7-needle slot-screens of 17 gauge steel; at 110 lb., its output was 192 t. Inside amalgamation was impossible, and the force of the discharge made the average mesh of the pulp much coarser than that passing the same size screen from ordinary stamps. When using finer screens, the pulp banked up and burst them; and when run at lower speed, there was no economy over the gravitation stamp. Consequently it was rejected.

But smaller types of steam stamp have recently been adopted in a few places, being efficient self-contained machines, easily installed and removed.

A Wood steam stamp working at the G. E. M. mine, Cherry, Arizona, mills at the rate of 10½ t. per 24 hours, and loses ½ lb. of iron per ton. In this machine, the frame is mounted on the foundation separately from the mortar, so as to reduce vibration of working parts. The pulp issues from

3 sides of the mortar. The engine or cylinder, with its valve mechanism, and the upper portion of the frame, are adjustable in height to the two upright stands by blocks or shims, in order to accommodate from time to time the wear of the shoe and die. The stem or piston-rod, the piston-head, boss and shoe weigh about 550 lb., and according to the drop, or rather the length of the direct stroke, which may be regulated at the will of the operator, the work of 3 1000-lb. gravitation stamps is claimed. The diameter of the shoe is 8 in., rod or stem $3\frac{1}{8}$ in., and piston-head $5\frac{1}{2}$ in. The power required is 60 to 70 lb., and may be steam or air. The valve mechanism consists of a simple piston-valve, operated by means of a tappet on the stem, engaging a bell-crank lever as the stamp-stem reciprocates, alternately opening and closing the ports. The action is similar to an ordinary engine, with the travel of the valve governed to meet variable requirements. The reciprocation is rapid, and 200 blows per minute can easily be obtained. The automatic feed is of simple design. The mill is grease-proof, and has a total weight of 4000 lb.

Another working at the Mammoth mine, Arizona, mills 7·86 t. of hard quartz per 24 hours through 40 mesh.

A Mexican Co. has three of them installed, as shown in Fig. 68.* They make 180 drops a minute of 10 to 11 in., and crush 8 to 10 t. each per diem through 12 mesh. Each consumes 16 t. of firewood per week, and about 4 gal. lubricating oil. Shoe and dies last 10 to 14 days, and screens 5 to 8 days. As compared with a stamp battery working on same ore, they give a more uniform product for concentration, affording less slime. In capacity, 3 Wood's slightly exceed 10 gravitation stamps. They consume more fuel per ton crushed, but no precise figures have been published.

The Tremain steam stamp consists of three stamp-stems, the upper ends of which terminate in pistons, working in cast-iron cylinders after the manner of the steam engine. These pistons are turned out of the solid forging which forms the stamp-stems, are $5\frac{1}{2}$ in. diam., and are fitted with 3 sets of piston-rings, making them steam-tight. The piston rods which pass

* En. & Min. Jl., March 31, 1900.

through the stuffing-boxes are 4 in. diam., therefore the steam pressure which is admitted under the piston to raise the stamp is confined to an area which is due to the difference between

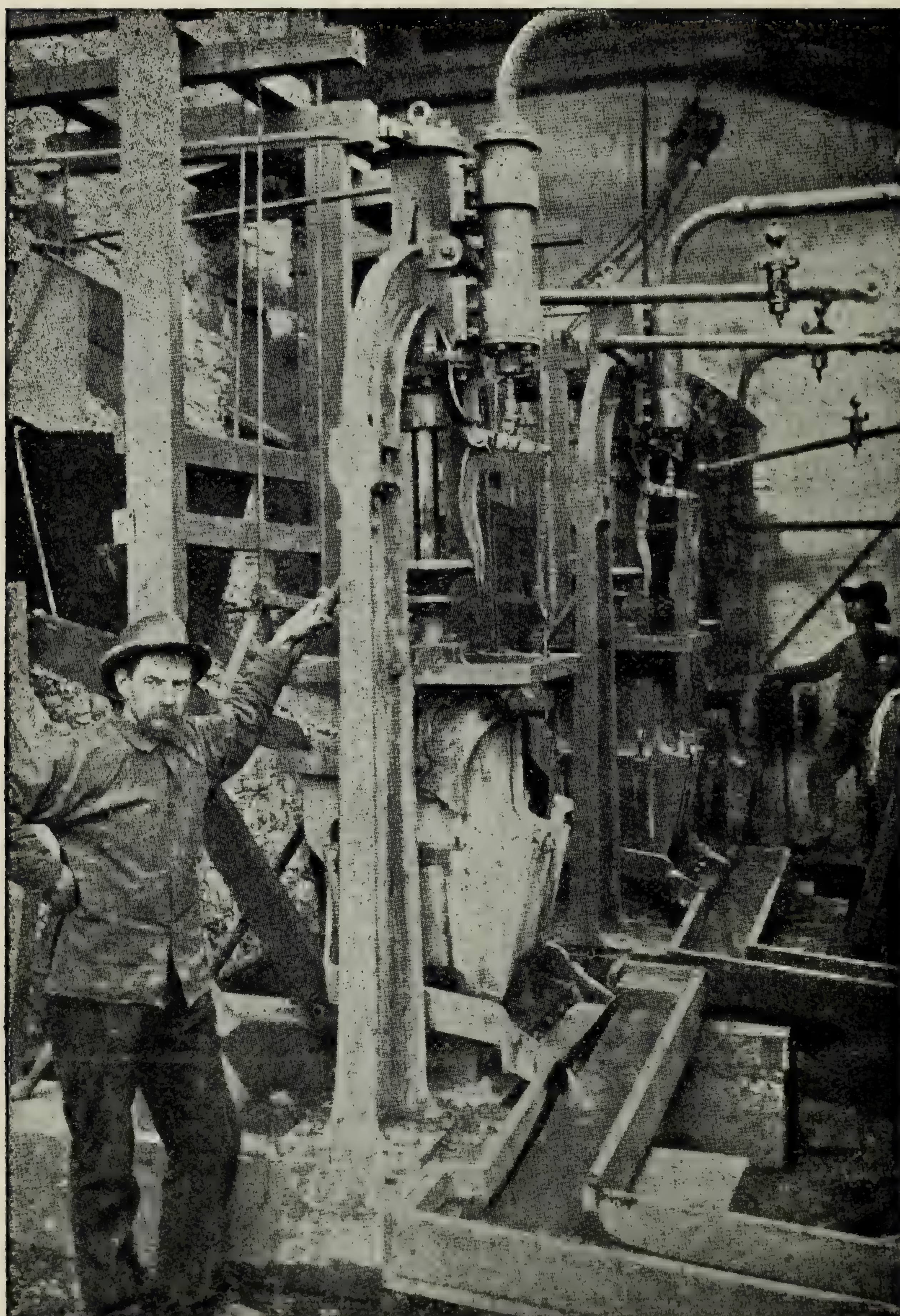


FIG. 68.—WOOD STEAM STAMPS.

the diameter of the piston and the piston-rod, amounting to an annular ring about $\frac{3}{4}$ in. wide; this area is small, but sufficient to quickly raise the stamps, the total weight of which is but 300 lb. Either piston in its travel toward the top of its cylinder passes a small steam-port, which admits the pressure to the valve mechanism, and moves the valve to its opposite position; the movement of the valve cuts off the admission of steam to the underside of the piston, and admits to the underside of its mate, at the same time connecting the top and bottom ends of the first-mentioned cylinder together. It thus allows the confined steam, which is holding the stamp up, to be expanded around the piston to its upper side, and, acting expansively upon the large area there encountered, to so energetically assist the 300 lb. stamp in its downward movement as to strike a blow upon the die equal to that of an 800 to 1000-lb. gravitation stamp. The pistons alternate with each other, and when the valve is moved back to again admit steam to the underside of the first-mentioned piston, it also connects the top side with the exhaust-port, so that the steam remaining after the blow has been struck is passed into the atmosphere. This arrangement makes it possible to use the steam expansively, and to obtain the same crushing effect with each 6-in. drop of the 300-lb. stamp as would be obtained with a gravitation stamp of 800 to 1000 lb. dropping 8 in. (depending upon the pressure used). A speed of 200 or more drops per minute of each stamp is attained. The machine is complete in itself, weighs but 3300 lb., and, being built entirely upon the mortar, requires no framework other than a substantial mortar-block. It is calculated to pulverise 8 to 18 t. of ordinary gold quartz through 40-mesh screen in 24 hours. The mill is provided with silver-plated lip-plates on the mortar, which take the place of the inside copper. The screening capacity is relatively large, there being about 540 sq. in. of screens used in the mortar, as against 475 sq. in. in the standard 5-stamp mortar; and, because of the very rapid movement of the stamps, a much greater agitation of the pulp in the mortar is kept up, and a much greater height of the screen surface is made available for discharge of pulp. The stems are free to

turn both ways, so as to regulate the wear of shoes and dies. The former are of forged steel, and the latter of cast iron. False bottoms are used to maintain an even discharge. A battery of Tremain mills erected by the Gates Iron Works, San Francisco, at the Regina Gold mine, is shown in Fig. 69.

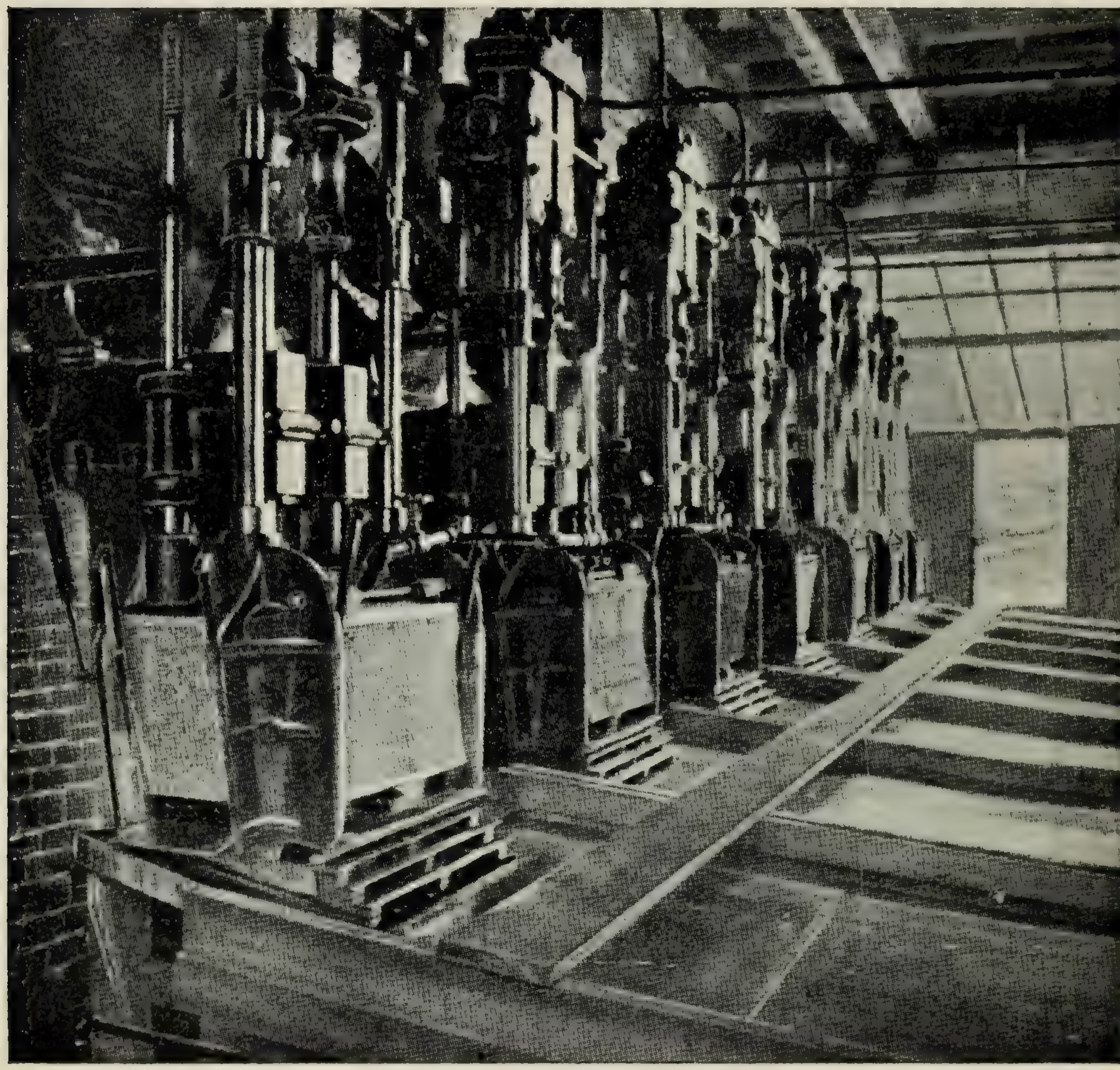


FIG. 69.—TREMAIN STEAM STAMPS.

A self-contained little plant well adapted for prospecting work or for small co-operative parties is turned out by the New Steam Stamp Mill Syndicate, Salford, near Manchester. The entire plant can be installed in a few days, embracing mill, timber foundation, copper tables, blanket tables, water-service pipes and steam boiler with mountings. It is made in 3 sizes, as follows :—

| Duty. | Blows per head per minute. | Weight of Issue | Weight of complete Plant. |
|---|---|---|---|
| 3 to 5 t. . . . | 250 | 12 cwt. | $4\frac{1}{2}$ t. |
| 6 to 8 t. . . . | 220 | 22 ,, | 6 t. |
| 10 to 12 t. . . | 200 | 35 ,, | 8 t. |

Good reports are given of its running in West Australia. An installation at Kanowna is shown in Fig. 70.

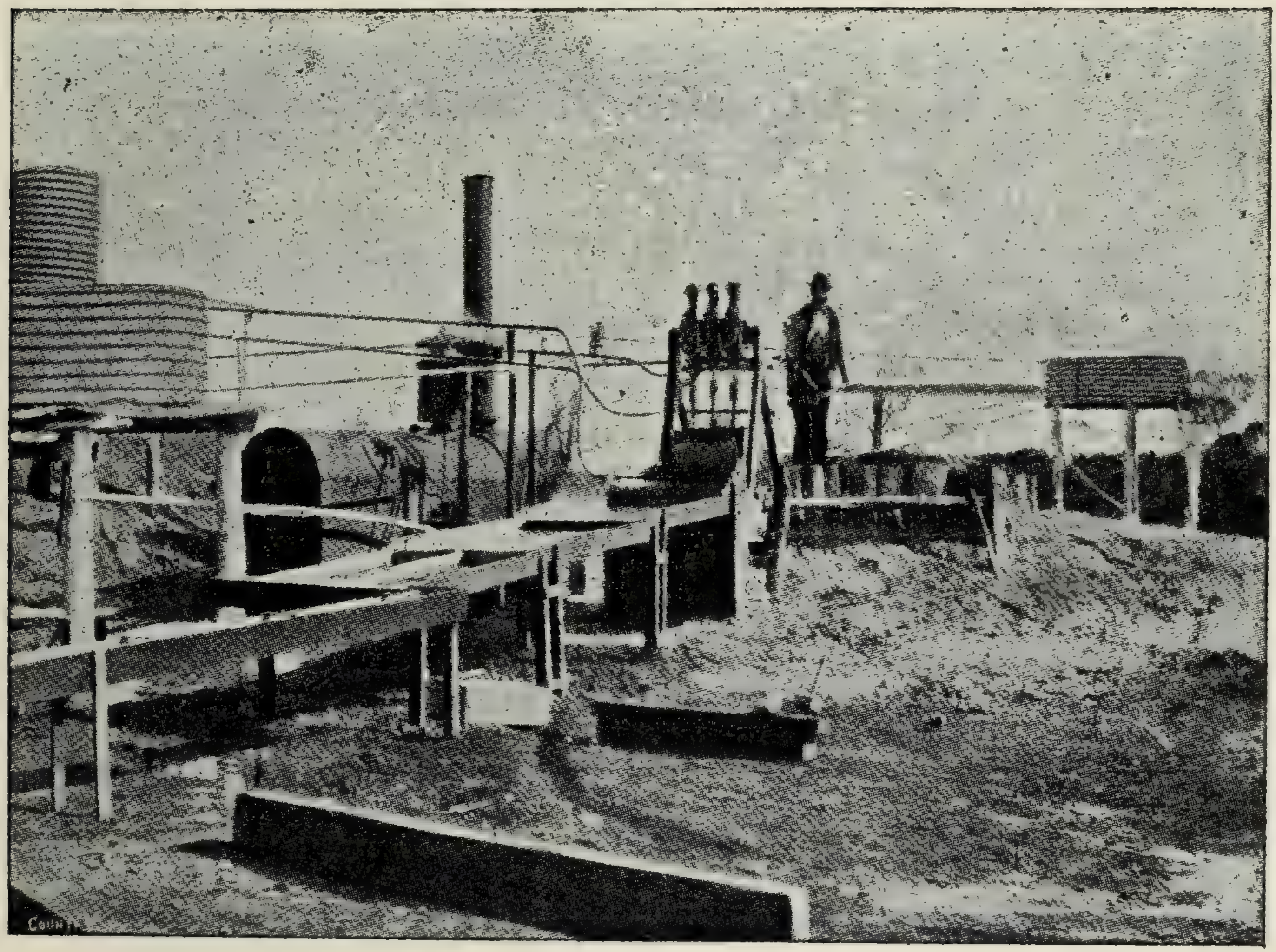


FIG. 70.—STEAM STAMPS.

# CHAPTER V.

## AMALGAMATION.

In preceding chapters, a description has been given of the machinery and methods by which gold-bearing stone is reduced or pulverised, in the presence of water, to a degree of fineness calculated to set free the gold particles. The next step is to collect and recover those particles, and for this purpose advantage is taken of the natural affinity which exists between gold and mercury, and of the ease with which their combinations can be broken up again without appreciable loss of either metal. This is called amalgamation. An amalgam is any alloy or union of metals in which mercury figures. Gold forms alloys with many metals, notably copper and lead, and this circumstance is availed of in other processes of gold recovery, but in those cases the union is much more intense, and subsequent dissociation of the two metals is much more difficult; in addition to which, the combining metal must first be brought to a liquid state, whereas mercury is already fluid at ordinary temperatures.

Since the earliest days of gold mining these facts have been known and utilised, and it is no exaggeration to say that even now three-fourths of the gold won from crushed ore is collected by means of mercury. The term collected aptly describes the operation. Amalgamation is not in any sense an extraction process, such as chlorination or cyanidation; it is simply a collecting and agglomerating of numerous particles into a mass which is handy of manipulation. It is essentially dependent upon two conditions—that the gold is in a "free" state; and that the particles are of such size and shape as to be able to bring their superior specific gravity into play, and

thus come to rest in contact with the mercurial surface presented to them. In other words, coarse clean gold alone is capable of economic recovery by amalgamation. Fortunately the bulk of the gold found in nature fulfils those conditions, and thus admits of the successful application of this simplest of metallurgical processes. At the same time, there are many circumstances demanding most careful study and consideration in order to obtain the best practicable results.

The chemistry of gold amalgamation is fully dealt with by Professor Louis,* who shows that, while a certain degree of dissolution of the gold in the mercury takes place, yet the amount retained in solution is very small, ranging from a trace to 10 gr. in the lb., and normally amounting to about 1 gr. of gold in 30 lb. of distilled mercury. At the same time, the gold retains 1 to $1\frac{1}{2}$ % of mercury, which is not eliminated till the gold is melted. In the ordinary sense of the word, "solution" seems an improper term to apply to amalgamation, because by dissolving amalgam in nitric acid, the gold particles can be recovered in their original form. However, this is an academic rather than an industrial question.

**The Gold.**—Quite apart from any environment or coating which may hinder access of mercury to gold particles, and thus prevent amalgamation, free gold is by no means all amalgamable. For instance, gold which has been thrown down as an amorphous powder from chloride solutions, by means of ferrous sulphate and other reagents, obstinately resists amalgamation, notwithstanding that it is quite pure. On this subject many investigations have been carried out by Professor Louis, who is inclined to suspect the occasional existence of this allotropic form of gold in nature. Its occurrence has, however, never been reported on anything like an industrial scale, unless the so-called "rusty" or "black" gold can be proved to be identical with it.

Some experiments made long since by Professor Egleston, showing that gold which had been hammered repeatedly on an anvil refused to amalgamate, have led to frequent repe-

* 'Handbook of Gold Milling,' p. 84.

tition of a charge against stamp batteries that they exert a similar effect, and thus cause some gold to escape amalgamation. It may safely be said that such a thing never occurs. In a properly fed mortar there will always be 1 to 2 in. of stone interposed between the shoe and the die. Further, there is a constant and violent flow of pulp to and fro, incessantly forcing away from the die all particles of matter as soon as their size allows them to be influenced by it. Any piece of gold having sufficient weight to resist the agitation, and remain quiescent on the die, may with confidence be trusted to need no amalgamation for its recovery. It would be impossible to lose it—except in the millman's pocket. The same may be said of those particles which are supposed to become studded and enveloped by fragments of crushed stone, and of those which are imagined to become abnormally buoyant by being flattened between the shoe and die. Suffice it to say that no one has ever yet obtained the least tangible evidence of such happenings, and they are, in fact, impracticable.

But there are at least two inherent conditions of gold which militate severely against amalgamation, viz. minuteness of particles, and envelopment in resisting media.

*Float Gold.*—The existence of what is called "float" gold is very real and very widespread, in fact, it is almost universal; but it is often overlooked because a substantial proportion of the gold contents of the ore under treatment is readily recovered, and returns a profit on the undertaking, consequently there is no stimulus to seek for the more elusive portion. Sometimes, however, the whole of the gold is in this microscopically fine condition, and visible particles are almost or even entirely absent. This is the case, for instance, with the Potsdam ore bodies of the Black Hills Dakota, which have yielded, and are still yielding, tons of gold; also with the ores of the well-known Waihi mine, New Zealand. Such gold will float almost indefinitely, that is to say, there is not sufficient mass in the particles to overcome the cohesion of the water, and resist the convection currents which exist in all volumes of liquid, even when apparently at

absolute rest: their specific gravity ceases to be able to exert itself. A few examples will help to impress this fact.

Cosmo Newbery, who was for many years metallurgical adviser to the Victorian Government, found gold at the rate of 42 oz. per ton in some "slimes," or impalpable matter that only coloured the water, from a New Zealand ore; and, in another instance, in the proportion of 20 oz. per ton in slimes escaping from Frue vanners.

J. H. Collins has described * a Hungarian ore which was treated in an excellent mill of the American type, with inside and outside plates, Frue vanners, and large settling pits for the slimes; the man in charge was skilful, the water was good, the plates were in perfect condition, and the effluent water went away apparently clean enough to drink; yet in that clean effluent water there escaped 55·07 % of the total gold contents of the ore treated. Estimating 20 tons of water used for each ton of ore milled, it would seem that every ton of water carried away about ·66 to ·75 dwt. of gold.

In some experiments at Lucknow, N.S.W., the author found that water from settled tailings, which had been allowed to stand untouched for three days and nights, was still appreciably cloudy, and this suspended matter, removed by filtration, assayed considerably higher than the ore originally supplied to the battery.

Vautin had gold suspended in perfectly still water—knowing the gold to be in a metallic condition before it was added to the water—for three months, and the metal was only satisfactorily separated by submitting the water to extreme centrifugal action; and in some of the ore from Kalgurli, W.A., he found the gold so finely divided that if it were placed in a bottle of distilled water, corked up, after 48 hours, some 60 to 70 % would still remain suspended—"it was, in fact, so fine that a mere breath would scatter it."

In very many instances, no doubt, the gold which resists the action of gravity is enveloped in grains of pyritous or other matters, so that its degree of sub-division is beyond the grasp of the human mind. Obviously this is a natural

* Trans. Inst. M. & M., i. 221 (1893).

condition of the gold, and not one induced by battery action or any other artificial means. In fact, it would hardly be possible to devise any mechanical contrivance for effecting such extreme trituration of a notoriously tough and ductile metal.

The shape of these tiny gold grains is most important as affecting their buoyancy, and would seem, from an experiment made by J. H. Collins,* to be such as is calculated to greatly favour it. He took some mercury which had been squeezed through an extremely fine cambric handkerchief, treated it with nitric acid, and what was left was a black powder. On examining the black powder under the microscope, he found that it was "composed of perfectly distinct scales, exactly resembling in form the scales got from that part of the amalgam that remained inside the handkerchief, only smaller. Every scale precisely resembled its larger brother. There was no indication that the gold had been dissolved in the mercury at all; it was simply mechanically mixed with it, and went out through the pores of the handkerchief, mechanically mixed still, and became visible on dissolving out the mercury and leaving the gold behind. 1-10,000th of an inch is large for a grain of this fine gold. You may measure some of these scales under the microscope, and while perhaps not more than 1-10th part as thick as they are in diameter, yet the diameter shall not exceed 1-10,000 inch. I am afraid to reckon how many millions of grains would be required to make 1 dwt. of gold of such fine particles as this."

*Coated Gold.*—"Coated" gold is not uncommon. In some cases the term "rusty" is well applied, the native gold particles being shielded by a film of iron oxide. In other instances the gold is rendered black by a jacket of manganic oxide of iron, or the envelope may consist of silica or a silicate of iron, while in the Transvaal it has been asserted that mineral oil, permeating the formation, has proved a serious obstacle to amalgamation. Even ores which have been roasted to drive off sulphur, arsenic, etc., are not always free

* Trans. Inst. M. & M., i. 224 (1893).

from the same drawback, as imperfect roasting may result in the gold being concentrated within a melted monosulphide of iron, or coated with a skin of magnetic oxide, or with tellurous acid or tellurite of iron. So pernicious is this coating of the gold that, even when the coat is thin enough to be transparent, the gold will resist amalgamation after weeks of immersion in mercury. Indeed, as is well known to millmen, the grease which may be rubbed on to a piece of gold by repeated handling will act as a bar to amalgamation. In all cases amalgamation is impossible without absolute contact between the mercury and some portion of the gold particle.

Many examples of gold coated with oxide of iron, and quite proof against attack by mercury, have been found in California, amalgamation ensuing immediately on rupture of the skin, even if only a very minute area of clean surface be thereby exposed. Sometimes the coating is so thin as to be semi-transparent, the metal being visible through it, yet its action is just as effective. Much of the reef gold found at Canoona, as well as the alluvial gold in Sharper's Gully, both in Queensland, is covered with manganic oxide of iron, the source of the gold in each case being a felstone-porphyry. A large proportion of the gold occurring in the gravels of the ancient (possibly Miocene) bed of the Macquarie river, New South Wales, is quite black, and will not amalgamate. In Venezuela and Brazil, the same thing occurs, the *ouro preto* (meaning "black gold") of the latter being of a dark purple hue, which E. M. Gibson thinks is due to its association with specular iron, the gold being probably discoloured by ferric oxide.

No chemical analysis of such ores as the preceding would give any idea concerning the condition of the gold or the existence of a coating on the metal, and only an actual trial will disclose it.

*Ores.*—Besides these conditions of the gold particles themselves, note has to be taken of the foreign matters associated with the gold in the ore under treatment. The presence of only clean sulphurets, especially the common cubical iron pyrites, does not materially interfere with amalgamation.

But it is rare to find auriferous mineral which does not contain a larger or smaller percentage of the readily decomposed cupreous, or arsenical, or antimonial sulphides, which, on decomposition, seriously affect the usefulness of the mercury, apparently by forming sub-sulphates with it, and adhering to its surface. They thus lead to a double loss—of mercury, which is sulphated and washed away, and of gold, which cannot reach the mercury through the film of sulphate.

Mactear encountered a South American ore consisting almost entirely of hydrated oxide of iron which, by most careful amalgamation methods, would not yield more than 35 to 40 % of its contents. The same feature is manifested at Mount Morgan, Queensland, and has been discovered by A. C. Claudet in some West Australian ores. Some experiments made by Johnson and Matthey on a similar ore from the latter place resulted in a maximum extraction of 35 %. In such cases, a roasting sufficient to de-hydrate the oxide of iron (below red heat) is a necessary preliminary to the action of mercury, after which amalgamation will recover 85 % of the value.

The presence of mispickel in the ore is by no means necessarily a bar to successful amalgamation. It would seem that, unless it is undergoing decomposition, it has no prejudicial effect. Thus at Lucknow, N.S.W., it was the author's practice to subject the almost solid arsenical pyrites (containing always some native antimony as well) which constituted the hand-picked ore, assaying from 25 to 500 oz. gold per ton, to stamping and amalgamation, so as to recover as much of the free gold as possible, and facilitate sampling of the remainder before disposal to the smelters. In no instance was any sickening of the mercury experienced, nor were the plates dulled.

A similar condition has been noted at Deloro, Canada, where mispickel predominates in the ore.* Here for some years the bromo-cyaniding process has been in vogue, but while operating satisfactorily on the fine gold, it has failed to take out the coarse gold. After a trial of amalgamation on a

* J. T. Donald, En. & Min. Jl., Dec. 3, 1898.

lot of 100 t., using a stamp mill and ordinary plates, and securing an extraction of 7 dwt. out of a total contents of 13 dwt., the plant has been newly installed, the intention being to treat the ore as if it were free-milling, recovering coarse gold by amalgamation, removing mispickel by vanner concentration, subjecting these concentrates to bromo-cyanide, and finally roasting them for collection of the arsenic. This is a particularly interesting case, showing a reversion to the old-fashioned practice of stamp milling and amalgamation after the long continued use, first of chlorination and latterly of the most complicated and "advanced" form of cyaniding. According to Manager Kerr, of the Belmont mine, amalgamation successfully extracts 60 % of the value of these low-grade ores, the only care necessary being to avoid excess of mercury.

**The Mercury.**—It is surely unnecessary to insist that pure clean mercury is as essential as free clean gold. Too much reliance cannot be placed on the purity of mercury as purchased from the merchant, and as it absorbs many other metals, such as copper, lead, zinc, antimony, and bismuth, its repeated use in the battery will leave it each time probably more contaminated.

*Purifying.*—Re-distillation does not effect a purification from the more volatile of the foreign metals, and they are the most objectionable, though it has been stated that a partial cleansing may be achieved in this way. The plan advocated is to collect the bulk of the mercury in one vessel, and then, before applying the finishing heat (see p. 266), to divert the condenser pipe into another receptacle, keeping the last 10 % of the distilled mercury separate from the remainder. This last portion is said to carry nearly all the impurities. Its further treatment can be taken in hand at leisure. Distillation at a low temperature in a current of superheated steam has been recommended; so has distillation under a layer of cinnabar (the sulphur of which combines with the foreign metallic bodies, and arrests them), or a layer of quicklime or iron-filings (which will absorb any arsenic or sulphur liberated),

the operation being conducted very slowly and in a capacious retort.

Simple agitation with air brings about oxidation to some extent of bismuth and other matters, but the action is only partial. Vigorous shaking with crushed lump sugar in presence of air, and subsequent filtration through stout blotting-paper with a small perforation in the point of the cone, will remove suspended impurities. Agitation with strong sulphuric acid for some days, or repeatedly with sulphuric acid and potassic chlorate, are also advocated. Digestion with hydrochloric acid or with ferric chloride removes zinc and tin, and heating with solution of mercuric nitrate dissolves foreign metals as nitrates. But the best plan of all, according to Prof. Louis, is to agitate with dilute nitric acid (1 of acid to 3 of water) for 24 hours, and allow to rest for a few days, drawing off supplies from below, and leaving the solution standing on it, adding acidulated water occasionally if nitrate of mercury encrusts the surface. Welton * has found very good results to follow washing the mercury in a strong solution of alkali obtained from wood ashes, where greasy or resinous contaminations have impaired its activity and brightness.

Pure mercury will break up into perfectly spherical globules, of uniformly bright surface, uniting at once when brought into contact with each other, and leaving no "tail" when allowed to run slowly down an inclined glass plate. It affords no black powder when shaken up with dry air in a bottle, and leaves no film on rough white blotting-paper.

*Flouring.*—Under agitation, and especially agitation with solid substances, such as occurs in a battery box, mercury readily breaks up into a multitude of tiny spherical specks. This is termed "flouring," from the metal's resemblance, when in that condition, to a white powder. It is aimed at in milling, so as to distribute the mercury among the ore. Each particle is then surrounded by a film of air or water, as the case may be, and, if not otherwise contaminated, on removal of that envelope of air or water, the globules will coalesce. So that

* Trans. Inst. M. & M., viii. (1900).

mere flouring is not in itself a serious detriment, unless the comminution is carried to such a degree that, like flour or float gold, the effect of gravity is destroyed, and the tiny globules get carried away in the effluent water.

*Sickening.*—But it is exceptionally rare for flouring to take place without there ensuing a fouling or contamination of the particles, which is known as "sickening," and in which state the mercury is quite inoperative, for the simple reason that contact with the gold particles is prevented just as much as when the gold is "coated." Grease is almost always present among the contents of the stamp mortar, being derived from lubricants and illuminants used both in the mill and in the mine. But a much more potent factor in sickening is due to the ore. Unctuous matters, such as clay and talcose minerals, rapidly affect the globules of mercury, covering them with an impenetrable film. Oxides of iron and of manganese, which are abundant in the produce of some mines, have a similar influence. Most common of all, perhaps, is the acid sulphate of iron arising from the decomposition of one or other of the almost ever-present sulphides,—copper, iron, and arsenical pyrites. Its injurious effect is singularly well illustrated by a group of mills in Colorado, situated on the same stream and using in turn the same water. The first of the series has a loss of mercury equal to 21·5 oz. per 100 t. milled, whilst the last loses 49 oz. The increase is on a graduated scale, except at the second mill, and there the loss is augmented by prolonged contact, so that it reaches 48·5 oz.

Steps are usually taken to remedy the evil, by addition of an alkaline substance to the milling dirt. Thus at many mills quicklime is introduced, in quantity roughly sufficient to absorb all sulphuric acid as fast as it is generated: for example, in one case a bucketful of lime is fed to each 10 stamps every 2 hours; at another, 1 lb. per stamp every 24 hours; and so on. Similarly, in barrel amalgamation of blanketings, quicklime or wood ashes—the former far preferable—is employed for a like purpose.

On the Rand, the use of lime has become general, its primary object in this case being to remedy the acid nature

of the ore in preparation for cyaniding. J. R. Williams* found it much more economical to add the lime in the mill, as it undergoes the grinding process of the stamps. Before the lime comes down to the slimes plant, practically all the lime that can be rendered soluble is soluble. He does not think the use of lime can be injurious to milling under any circumstances, but a mill manager using lime for the first time will find that the amalgam on the plates becomes very considerably hardened. He was called in to investigate one or two such cases, in which they said they were catching no gold on account of the use of lime. But there was one other very interesting result, and that was that the tailings were, if anything, a little lower in value, and therefore they were really losing no gold. On cleaning their plates at the end of the month, the output showed no falling off as had been supposed.

The injurious action of iron salts has been remarked in many cases.

Bismuth and antimony are specially active in creating a sickened condition of the mercury. The former is not very common, though perhaps, if specially sought for, its presence would be disclosed more often than is suspected. But antimony is a very frequent accompaniment of arsenical pyrites. When existing as stibnite, it is particularly objectionable, and may altogether preclude adoption of amalgamation methods for recovering the gold. As native antimony, it seems to do comparatively little harm, but it amalgamates readily (contrary to the text-books), making a very bulky clean-up, and necessitating more frequent rubbing of the plates. At Lucknow, N.S.W., it is an almost constant ingredient of the ore, and, in the author's experience, it was unwelcome only for the reasons given ; but perhaps the calcite (lime) gangue helped to neutralise any trouble it might otherwise have caused.

Sulphur and arsenic, when in a free state, are most injurious to mercury, dulling its surface in a marked degree. The vast majority of ores carry a certain proportion of iron and arsenical pyrites, the latter being generally the more auriferous, and if they are allowed to undergo decomposition while being

* Jl. Chem. & Met. Soc. S. Africa, Apl. 1898.

milled, much sickening will result; but in the absence of decomposition, arsenical pyrites is innocuous, as already described.

Tellurium in the form of tellurides is another insidious enemy to amalgamation, and, like antimony and bismuth, is often present when not supposed to be. The Potsdam "refractory" ores of the Black Hills, and the almost unique deposit of Mount Morgan, have been worked for years, and very many tests of them have been made; but it was not until quite recently that tellurium was discovered in both, satisfactorily accounting for the failure of all attempts to amalgamate their gold.

**The Water.**—The influence of water in the process of amalgamation may be exercised in two ways, physical and chemical.

The effect of impurities suspended in the water, as for example clay and soil generally, is to coat the mercury globules as with a film of grease, preventing intimate contact between gold and mercury surfaces. Chinamen and Mexicans, the best amalgamators in the world, carefully avoid muddy water, and even water in which bathing or clothes-washing has been done, and will take an infinity of pains to procure a pure supply. No wonder that many of the attempts to amalgamate in water of the consistency of pea-soup have failed.

Matters in solution may so increase the density of the water as to greatly exaggerate the buoyancy of the gold particles, and thus augment the loss of float gold. The saline waters forced upon the millman in some parts of South America and West Australia are of this class. Most of the available battery water in the latter contains 20 to 30 % of saline matters, and has a sp. gr. of 1·17 and upwards; sodium chloride predominates, but much magnesium and calcium salts is also present. At the Hampton Plains Estate, with highly saline water, the extraction only reached 30 to 40 %. It appears "that the loss was due to the high specific gravity of the water; the gold was very fine, and apparently went away in suspension." *

* Vivian, Trans. Inst. M. & M., v. 338 (1897).

Similarly, in operating on the ores of the Guanaco district, Chili, in which the gold is never visible, but is as fine as flour, and where sea-water has to be used, a loss of 10 to 20 % of the gold contents was experienced,* and this was clearly due to the fine gold floating away.

Other matters in solution, such as often occur in mine waters, and in the effluent water from stamp batteries, especially sulphate of iron from decomposing pyrites, operate chemically by attacking the mercury globules, converting them superficially into sulphates and subsulphates, and thereby destroying them as amalgamating media. Fortunately, as has been already described (p. 211), this may easily be combated by addition of quicklime, to the water before use if feasible, or to the battery during use if not. At the Bonanza mill,† S. Africa, it has been observed that the use of lime-water circulated from the slimes plant (cyanide process) in the battery has incidentally "tended to promote the amalgamation of the finest particles of gold in the mill, and so secured a direct benefit to the general output, since if the gold so caught were to pass on to the slimes plant, only 83 % of it would be secured."

On this question of acidity or alkalinity in the battery water, there has been much argument of late.‡ It is stated by D. E. Powrie, as the result of direct experiment lasting over 6 months, that nearly 5 % more gold was caught when the battery water had an acid reaction than when it was alkaline, and that the effluent slimes contained 9 % less value. Probably these results, if really true, will be found to be due in this instance to some peculiar condition which in previous investigation has been overlooked, because it is contrary to the conclusions arrived at by every other experimenter on the field. The general concensus of opinion is in favour of alkaline water. The existence of one dissentient case, however, is sufficient to show that the subject is still worthy of further study with minute attention to detail.

* G. M. Barber, Trans. Inst. M. & M., v. 101 (1897).

† Report for 1899.

‡ Jl. Chem. & Met. Soc. S. Africa, April 1899.

**Temperature.**— Amalgamation is designated as a chemical process, and it is a well-known rule in chemical reactions that they are favoured by heat, yet wide differences of opinion exist among millmen as to the most desirable temperature to be maintained in battery water.

Rickard, while admitting that gold amalgamation is assisted by heat, yet asserts that below 212° F. the effect of a rise in temperature is so slight that it is not worth seeking.

Prof. Le Neve Foster recorded at Pestarena a 3·1 % greater extraction of gold (by a species of pan amalgamation chiefly) in winter, when the water temperature averaged 39·4° F., than during the summer months, when its mean was 52° F.

So also in the elevated regions of Colorado, where the winter cold is extreme, the millmen prefer water which is just warm enough to avoid freezing on the plates, the reason alleged being that cold stiffens the amalgam, and makes it more retentive of gold, white heat softens it, and makes the mercury sweat out of the plates and roll down into the traps.

At the Homestake mills, Dakota, where the climate much resembles that of Colorado, trouble used to be taken to maintain the battery water at a temperature of about 70° F., by leading the engine exhaust pipe through the supply tank in winter, and at that temperature no complaint was made of the mobility of the mercury, or an undue sweating of the plates; but more recently it has been stated by Superintendent Grier * that amalgamation is more complete at 50° F. than at 60° to 70° F.

In Norway, Daw † finds no difficulty with amalgamation in the summer months, but in winter, when the water is drawn from beneath the frozen surface of the reservoirs, the consumption of mercury is much increased, by its separating out in large globules and rolling away. To enable amalgamation to be satisfactorily carried on during the cold months, live steam is charged into the main water-pipe until the temperature of the water reaches 50° F., the lowest degree at which the results

* En. & Min. Jl., Jan. 29, 1898.

† Trans. Inst. M. & M., v. 218 (1897).

are satisfactory. At this temperature, the mercury loss rarely exceeds ·297 oz. per ton treated, while at 41° F. the loss is 1·785 oz. per ton. This last is, of course, a most extravagant figure, costing (at 2*s.* per lb. of mercury) 2·963*d.* per ton milled. The extra cost for fuel for warming the water amounts to 2·679*d.* per ton milled, which, added to the normal mercury cost of ·494*d.*, totals more than the gain, so that, unless a reduction in loss of gold is also effected, the outlay on warming the water does not seem to be justified.

That a low temperature lessens the cohesive power of mercury, causing globules to sweat out and run away, especially when sulphides are present in the ore, is confirmed by W. F. Drake,* who finds that the water should at least be above 50° F., and that the maximum recovery of gold takes place between 80° and 90° F.

In a Transylvanian mill, where the summer temperature is 80° to 90° F. and the winter only 34° to 36° F., according to Von Dessauer,† theoretical and actual extractions vary little; but the amalgam carries 25 to 35 % gold in warm weather, and only 7 to 12 % in cold; and during spring and autumn, the stiff amalgam formed on frosty nights gives up much of its mercury in the day.

Other examples are quoted ‡ where a 90 % recovery is got at 80° F., and only 80 % at 60° F., the ore carrying some pyrites and galena; and where the mercury fed per oz. of gold is much less at 90° F. than at lower temperatures.

Manager McDowell § at the Rose Deep mill, in the Transvaal, expresses a preference for about 90° F., though he gets good results at 76° F. and at 100° F. In his opinion, actual temperature is not so important as regularity, and he finds that changes of temperature are the principal source of trouble.

This latter opinion is endorsed by J. A. Sanborn,‖ who considers that the millman who regulates his work by the con-

* Min. & Sci. Press, Dec. 3, 1898.

† Jl. Chem. & Met. Soc. S. Africa, Apl. 1898, p. 52.

‡ 'Mineral Industry,' vii. 352 (1899).

§ Jl. Chem. & Met. Soc. S. Africa, Apl. 1898, p. 41.

‖ En. & Min Jl., Apl. 2, 1898.

dition of his outside plates should get equally good results "whatever the temperature of the water, within the bounds of reason and uniformity."

F. F. Sharpless * records an experience, where as summer advanced the amalgam on the plates became increasingly hard and persistently adherent, notwithstanding constant addition of fresh mercury, until the temperature of the water reached 83° F., when operations had to be suspended because of excessive loss of gold in tailings. As at the Homestake mill, these plates are electro-silvered. But the last 8 ft. of the apron plates were very old, and from portions of them all the silver had been worn, leaving practically plain copper. On these portions, amalgamation proceeded regularly and satisfactorily at all temperatures.

The contradictions indicated by the foregoing variations of practice are perhaps after all more apparent than real. Account must be taken of differences in the milling dirt, in the gold, and in the water.

Some gangues exert a notable hardening influence on amalgam, and this is particularly the case with calcareous matter. In the author's own experience at Lucknow, N.S.W., where calcite is abundant in the stone, the amalgam is so hard that its removal always needs a chisel, even after prolonged steaming ; hence warm or even hot water will cause no sweating of the plates, and in this instance, at any rate, the objection named has no existence. Doubtless these conditions are duplicated elsewhere.

The decomposition of sulphides proceeds much more rapidly in summer than in winter, so that the ore, and in many cases the water also, then presents inferior opportunities for amalgamation, and causes more sickening of the mercury, and more scum or verdigris on the plates. This entails more labour for the mill hands, and, as is the way with most workmen, anything that gives trouble is condemned. In all likelihood, the addition of quicklime to the boxes in the case of those Colorado batteries where cold weather is preferred for amalgamation would evoke another opinion.

* En. & Min. Jl., Aug. 13, 1898.

In the Pestarena example, it is highly probable that not only sulphates in solution may play a part in the percental extraction, but also suspended matter, as the water is drawn from glacier-fed streams, and these are notoriously muddy. Some figures derived from the official report of 1898 seem to to throw light on the matter. They are as follows :—

| — | Fine Gold per Ton. | | Extraction. | Mercury Loss. | |
|---|---|---|---|---|---|
| | Contents. | Extracted. | | Per ton milled. | Per oz. of Bullion. |
| | dwt. | dwt. | % | dwt. | dwt. |
| January . . . . | 21·50 | 18·29 | 85·0 | 139·41 | 123·00 |
| February. . . . | 20·75 | 17·87 | 86·1 | 130·66 | 123·00 |
| March . . . . | 20·50 | 17·46 | 85·2 | 131·25 | 126·87 |
| April . . . . | 18·50 | 15·75 | 85·1 | 83·32 | 90·16 |
| May . . . . . | 19·25 | 16·42 | 85·1 | 84·48 | 86·94 |
| June . . . . . | 16·75 | 13·98 | 84·0 | 136·50 | 162·29 |
| July . . . . . | 14·75 | 12·33 | 83·7 | 110·25 | 145·54 |
| August . . . . | 15·00 | 12·70 | 84·6 | 148·15 | 195·13 |
| September . . . | 18·00 | 15·21 | 84·5 | 148·15 | 164·22 |
| October . . . . | 9·75 | 8·02 | 82·3 | 146·09 | 301·39 |
| November . . . | 7·00 | 5·37 | 76·7 | 139·92 | 414·74 |
| December . . . | 7·25 | 5·71 | 78·7 | 165·09 | 466·90 |

From these, it will be seen that, in April and May, the extraction was a fraction better than it was in January, while the loss of mercury per oz. of bullion won was about 50 % less. In June, July, August and September, working on lower grade ore, the extraction was only a trifle less. Contrasting the July and January figures, there is a shortage in the former of but ·21 dwt. per ton, even if the same proportionate extraction be looked for from 14·75 dwt. stone as from 21·50 dwt., which is manifestly unfair ; and moreover, the loss of mercury has gone up by leaps and bounds, due, as may reasonably be supposed, to sulphating or to muddy water. The best results all round are obtained in April and May, just those months in which the benefit of warmer water would be felt without the drawbacks of high summer, when the glacial water would be

at its foulest. The figures relating to the final three months of the year do not compare, because there is such a great discrepancy in the grade of the ore.

The marked difference in the effect of temperature according as the plates are of plain copper or electro-silvered is most important, and points to a distinct superiority for the former.

Gold which is in very fine particles and of high standard hardens amalgam, according to the experience of E. R. Woakes and of A. L. Collins,* making it difficult to remove even with scrapers, and this when milling with water practically constant at 70° F.; but it is perhaps premature to accept this statement without knowing more about the ore. Woakes admits that "the gangue contains a good deal of lime," though he doubts "whether this has much bearing on the point." Others have found that it is a potent factor. Moreover, gold in fine particles produces an amalgam containing much more mercury, which should have a softening influence.

**Inside Amalgamation.**—This term is applied to the amalgamation which is effected inside the battery box. It is accomplished in two ways, viz. by adding mercury to the pulp periodically, and by fitting the mortar with amalgamated plates.

*By Mercury.*—The rational practice is to furnish such a supply of mercury as will maintain the amalgam on the outside plates in that condition which, without being too hard and dry, does not show any weeping of mercury globules. The quantity of mercury supplied, assuming there be no cause for its destruction, should be directly proportioned to the amount and grade of gold present, or in other words, it will depend on the richness of the ore and the rate of crushing.

Rickard † gives the normal feed in the North Star mine, California, at $\frac{1}{2}$ to 2 oz. per battery per hour; P. R. Robert ‡ says $\frac{1}{8}$ oz. per ton per hour.

* En. & Min. Jl., Sept. 24 and Oct. 8, 1898.

† 'Stamp Milling of Gold Ores,' p. 44.

‡ Trans. Inst. M. & M., v. 153 (1897).

At the Hidden Treasure mill, Colorado, the millman has access to the inside plate, and regulates the mercury supply by the feel of its surface; upon an average, $\frac{1}{2}$ thimbleful is added every hour, which is about equivalent to 1 oz. mercury per oz. gold present in the ore.

In the four mills of the Homestake Co., Dakota, the figures vary considerably; while the three establishments running on about 5-dwt. stone add $1\frac{3}{4}$ oz., 2 oz., and $2\frac{1}{4}$ oz. mercury per oz. of gold respectively, the fourth, crushing $1\frac{1}{2}$ to 2-dwt. dirt, exceeds $2\frac{1}{2}$ oz. per oz.

At Lucknow, N.S.W., the practice is to have a tray of tiny bottles, each containing an hourly charge for one battery; the normal provision is $1\frac{3}{4}$ oz. mercury per oz. gold.

Daw * gives 3 oz. to the oz. for a Norwegian mill operating on 6 dwt. ore.

Many of the Transvaal mills make no use of mercury in the mortar.

Among Victorian examples, the Star of the East, which has no inside plates, introduces, according to Rickard, a teaspoonful per box every 2 hours, and taking his estimate of the teaspoonful at 4 oz., this would mean over $5\frac{1}{2}$ oz. per oz.; elsewhere he values the teaspoonful at 2 oz., which seems more reasonable. The Britannia United contents itself with 1 oz. per oz.; the Fortuna but little exceeds $\frac{3}{4}$ oz. per oz.; the South St. Mungo furnishes well over 2 oz. per oz.; the New Chum, $1\frac{2}{3}$ oz.; and the Harrietville, with no inside plates, and the abnormal feature of end discharges in addition to the usual screen area, is credited with the exceptional figure of over 5 oz. per oz.

There is great diversity of opinion on the merits of introducing mercury into the mortar. Obviously the metal must soon become broken up into a multitude of exceedingly fine particles, each one of which is thoroughly exposed to sulphating or other injurious action of ore and water, whereby its usefulness may be impaired or even destroyed, unless ample provision be made to correct this. It would seem that under the most favourable circumstances, but little actual

* Trans. Inst. M. & M., v. 218 (1897).

effective service can be rendered by the mercury while inside the battery, because, though amalgam can generally be collected from the boxes, it will be found that practically all the gold so caught is comparatively coarse, and could not travel far, even if it would be ejected from the mortar at all. It is also practically a universal experience that inside amalgam retorts a higher percentage of gold than outside. This is a distinct proof of the coarser size of the gold, the retort percentage diminishing in exact proportion to the dimensions of the gold particles absorbed. Therefore a more liberal allowance of mercury must be fed to ore carrying very fine gold. Where the gold is very coarse, inside amalgamation is dispensed with entirely; thus at the Port Phillip mill, Clunes, Victoria, about 45 % of the gold was picked out of the mortar in clean pieces; and at the Phœnix mill, Otago, New Zealand, 61 % was similarly recovered: the mere specific gravity of the particles suffices for their arrest. It is clear also that the bulk of the mercury speedily finds its way out of the box with the pulp issuing through the screens. This is perhaps the most rational explanation of the practice, for by this means the outside plates receive a constant replenishment evenly spread over their whole surface, without the attention that they would otherwise demand; and on this ground, if on no other, the custom is likely to continue so long as amalgamation is in vogue.

*By Plates.*—There are two positions in which inside plates may be adjusted to a mortar, viz. as a strip along the back beneath the feed opening and similarly along the front under the discharge. In Figs. 31 and 32 a number of examples of mortar boxes are shown, and the position of the plates is indicated, an instance being also given of provision for catchment of amalgam at the end of the box, though this is not strictly speaking an amalgamating contrivance. In Fig. 71, the arrangement of front plates is exemplified in greater detail: A illustrates the chuck-block with rounded surface; in B, it is plain; while C has no chuck-block, but the screen-frame carries a strip of copper-plate instead. In all cases, the reference letters indicate—*a*, mortar; *b*, chuck-block; *c*, copper

plate ; *d*, screen-frame lower bar ; *e*, screen. These constitute the principal variations of type, but hardly any two mills have exactly the same practice.

Among Californian examples, the North Star mill adopts pattern B, the plate being electro-silvered, 48 in. long, 5 in. wide, and $\frac{1}{4}$ in. thick, screwed to the chuck-block, and sloping inwards at an angle of 45°. The neighbouring Empire battery has a plain copper plate, 50 in. by 4 in., similarly fixed. In Amador County, pattern A is preferred as giving a more rapid discharge ; at the Gover mill, the chuck-block is 8 in. deep, and the copper plate is 6 in. wide and the full length of the mortar ; at the South Spring Hill, 6-in. and 7-in. chuck-blocks are used, according to the wear of the dies ; at the Wildman,

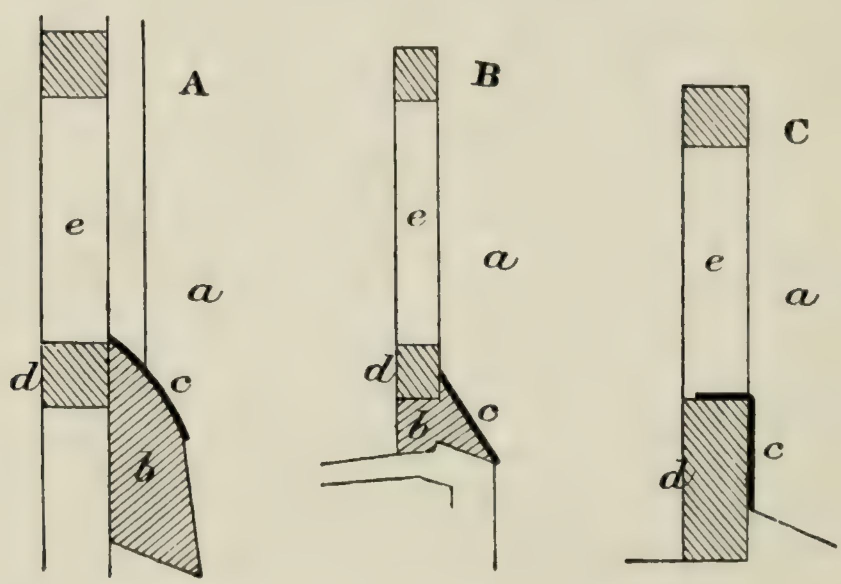



FIG. 71.—INSIDE PLATES.

they range from 4 in. to 7 in. ; and at the Kennedy, the chuck-block is made in 4 sections of $1\frac{1}{4}$ in. and 1 of 2 in., the latter alone carrying a plate.

In Gilpin County, Colorado, where inside amalgamation is carried to an extreme point by having unusually roomy mortars and deep discharge, both back and front plates are employed, of plain copper, and 54 in. long. The former is 12 in. wide, and placed at an angle of 40°, while the latter is 6 in. wide and almost vertical.

The Black Hills mills, Dakota, present some variety. In the Homestake battery the plate is of plain copper, 5 in. wide ;

that on the 7-in. chuck-block, used when shoes and dies are new, is of pattern B ; that on the 5-in. block is of pattern A, and mounted on thicker wood, so as to bring it nearer the die. The Columbus battery has a back plate 8 in. wide, and a front one 5 in. wide, on a 5-in. chuck-block, and though the mortar is only 12 in. wide, there is no scouring of these plates, notwithstanding a duty of 3½ t. per stamp.

According to A. C. Claudet,* the Mesquital mill, Mexico, has a back plate 50 in. by 6½ in., and a front plate 50 in. by 4½ in.

At the Lucknow battery, New South Wales, an amalgamated copper strip, about 4 in. wide, covers the bottom bar of the screen frame as in C. Trial was made of a back plate, but it was soon cut away.

In the Transvaal quite a number of batteries have no inside plates at all, and among those where they are used the back plate is more general than the front. The chuck-plate, when adopted, is mostly of A pattern. The City and Suburban has a 9½-in. back plate, and chuck-plates of 3 in., 4½ in. and 6 in., to suit the wear of the dies ; Crown Reef has 9-in. back and 6-in. front plates ; Ferreira, 11-in. back and 3¼-in. front ; Geldenhuis, 11-in. back, no front ; Jubilee, none ; Jumpers has discontinued inside amalgamation, and the result has been very satisfactory ; May Consolidated has none ; New Heriot, no back plate, but a 3½ to 6-in. chuck-plate ; New Primrose, 7-in. and 12-in. back, no front ; Robinson, 12-in. back, and 3-in., 4-in., 5-in. and 6-in. front ; Simmer and Jack, none at all ; and United Main Reef, ditto.

The Victorian practice is to avoid inside plates.

*Results.*—One of the strongest claims made for inside amalgamation is that it is a preventive of stealing. This is quite imaginary. The millman who intends to steal can do so from the inside as well as from the outside ; indeed, in many batteries, access is purposely given to the front plate for judging how much mercury to feed. Another potent advocacy for the practice is that the gold should be caught as soon as

* Trans. Inst. M. & M., iii. 347 (1895).

possible, lest it escape into the tailings. As already remarked, the gold which is captured in the box is always the coarsest, and if the outside plates are incapable of arresting it, their chances of catching the finer particles must be remote. Without involving delays, it is impossible to give the inside plates any attention, except at clean-ups, and if they are multiplied, milling time is lost. It may be questioned whether the amalgam accumulated on inside plates would not be more useful on the outside plates in helping to make them more receptive, it being a well-known fact that plates are more active as they become more loaded with bullion. At any rate, inside plates seem to be at a discount in the most progressive field in the world, and the following figures, taken from the published statistics of the mills quoted above, furnish an apparent justification, thus :—

| Mill. | Inside Plates. | Total Recovery by Amalgamation. | Total Value. |
|---|---|---|---|
| | | % | dwt. |
| City . . . . . . . . | Back and front | 54·75 | 8 |
| Crown . . . . . . . | Do. | 57·75 | 8 |
| Ferreira . . . . . . . | Do. | 60·00 | 21 |
| City . . . . . . . . | Back only | 56·50 | 7 |
| Geldenhuis . . . . . | Do. | 64·00 | 7 |
| Jubilee . . . . . . . | Do. | 70·00 | 10½ |
| New Heriot . . . . . | Front only | 62·00 | 9¼ |
| New Primrose . . . . | Back only | 61·00 | 6¾ |
| Jumpers . . . . . . . | None | 68·00 | 8½ |
| May . . . . . . . . | Do. | 66·00 | 7¼ |

Admitted that, in the Transvaal mills, leaching methods of treatment for tailings have met with great success, and tended, perhaps, to reduce the importance of amalgamation, yet the fact remains that amalgamation still affords in every case more than half, and sometimes more than two-thirds, of their total product, and would certainly not be neglected by such pre-eminent mill managers as there hold control. Hence their methods are well worthy of attention.

Of total amalgam, the proportions collected from inside the mortar in a number of mills are as follows :—

| | | | |
|---|---|---|---|
| California : | % | New Zealand : | % |
| Empire | 75 | Premier | 60 |
| Gover * | 79 | Norway. | 15 |
| North Star † | 66 | Queensland | 33 to 50 |
| South Spring Hill. | 55 | Transvaal : | |
| Wildman | 81½ | City | 17½ to 22½ |
| Colorado : | | Crown. | 34½ |
| Gilpin County | 66 | Jubilee | 15 |
| Dakota : | | New Primrose | 14 |
| Homestake | 50 | Victoria : | |
| Terra | 70 | Harrietville | 33 |
| Mexico : | | New Chum ‡ | 57½ |
| Mesquital | 70 | New Normanby § | 80 |
| New South Wales : | | Oriental | 50 |
| Lucknow | 30 | Pearl | 65 |

On this subject, W. McDermott ‖ says that with fine gold, particularly if the gold be of low standard, "it is often difficult to keep the inside copper plates in order, and the bother connected with them in cleaning up is compensated for by no advantage, so that it is better to use outside plates alone; this has been the experience in a number of mills, including, for example, that of the Montana Company, where inside coppers are discarded for outside amalgamation."

**Outside Amalgamation.**—The use of mercury and plates inside the battery has its parallel in outside amalgamation, where the copper-plate table is often supplemented, and sometimes replaced, by mercury wells and riffles.

*By Plates.*—The principle underlying the outside amalgamated plate is to provide an abundant superficial catchment area, over which the pulp is passed in a shallow stream of such consistence and at such speed as will afford a maximum opportunity for the gold particles to assert their gravity

* Retorts 37% bullion. † Retorts 40% bullion.
‡ Retorts 46% bullion. § Retorts 70% bullion.
‖ Trans. Inst. M. & M., i. 249 (1893).

and come into contact with it. The pulsation created by the stamps results in the pulp being discharged from the box in a succession of wavelets ; and the friction between the surface of the plate and the pulp immediately in contact with it, rendering the flow less rapid there than in the remainder of the stream, causes an incessant interchange of positions among the particles. These features are relied on to bring every amalgamable particle into actual contact with the plate at some period of its travel, and its arrest at the moment of contact is facilitated by the roughened surface and mercurial impregnation of the plate.

*Preparing Plates.*—There are two kinds of plates in common use, known respectively as plain copper and electro-silvered. The former is simply amalgamated copper, while the latter has an electro-deposit of silver before being amalgamated. A third kind, adopted in rare instances, is made of Muntz metal (an alloy of 60 % copper with 40 % zinc).

In either of the first two cases the sheet copper should be of the purest brand that can be got, as freedom from foreign metals is of vital importance in obviating spots caused by corrosion, due to local galvanic action being set up. A usual thickness is $\frac{1}{8}$ in., and at many mills $\frac{1}{16}$ in. is preferred for the sake of economy in first cost, a set of plates representing considerable value ; but this is a great mistake, because thin plates cannot be counted on to keep that level surface which is essential to their efficiency ; and while $\frac{1}{4}$ in. is not too thick for inside work, $\frac{3}{16}$ in. is a fair figure for outside plates. The sheets must have a perfectly true rolled surface, without buckle or dent of any kind.

The first stage in preparing the copper plate is to subject it to an annealing process by applying heat. The plate may best be laid on a sheet of iron, supported at some distance from the ground, taking care that the intended working face of the plate is kept uppermost. A fire of chips and shavings is kindled beneath, and the heat is maintained and distributed till every part of the plate will char paper thrown on it. This opens the pores of the metal, as it were, and makes it more absorptive of the mercury to be applied. With thin plates,

buckling is likely to occur, and must be remedied when the plate is cold by subjecting it to a hammering on a flat surface, using no more force than is actually necessary, and protecting the plate with a slab of wood to receive the blows.

Annealing being finished, the plate is transferred to the table, and the coating of oxide, which will have formed on the working face, is removed by wetting it and scattering all over it some fine, clean river sand (tailings are objectionable), and scrubbing with a block of soft wood till the whole surface shows the true copper colour. This done, the surface is freed of sand and moisture, and rendered quite clean and smooth by rubbing with fine emery-cloth. Then follows the actual amalgamation by rubbing in mercury, which is known to be quite pure, until the surface is uniformly bright and white. Mixing a little mercury at a time, by squeezing it through a shammy (chamois) leather, or strong stout calico, with sal-ammoniac moistened with clean water (distilled water is best —mine-water, mill-water and boiler-water are to be avoided), and about 10 times as much pure river sand, the application may commence from one corner, using a soft article for a rubber, such as blanket or calico, and continuing the process over the whole area. The surface is then washed quite clean, and left for 24 hours, during which time a scum or discoloration will manifest itself, due to slight oxidation of the copper. This is removed by applying a little weak ($2\frac{1}{2}$ %) solution of potassium cyanide.

The amalgamation of the plate is now completed, that is to say, the copper has become superficially saturated with mercury, and taken on the appearance of frosted silver. But its activity in retaining gold particles is not well marked in this condition, and may be very much increased by "setting" it with some gold amalgam from a previous clean-up, if obtainable, or, in default of that, with a purposely prepared silver amalgam. While the surface of the plate is kept moistened with the dilute potassium cyanide solution, the gold or silver amalgam in a limpid condition is vigorously applied, by the aid of blocks of pure rubber measuring about $5 \times 2\frac{1}{2} \times 1$ in., until the whole has been thoroughly coated.

There are several ways in which silver amalgam can be prepared, but the simplest of all is to procure some silver foil, in quantity sufficient to allow of about $\frac{1}{4}$ oz. per sq. ft. of plate, and mix it with such a quantity of mercury as will make a soft semi-fluid mass.

When silver foil cannot be had, a substitute may be found in silver coin, but its impurities must first be eliminated. This is done by gently warming some dilute nitric acid in a porcelain dish, and dissolving the coin in it. When the metal has disappeared, the solution is slowly evaporated to dryness in a water oven, and then the residue is heated till it begins to fuse, and the colour changes from bluish to a blackish-grey, which shows that the soluble nitrate of copper has been converted into the insoluble oxide. The mass is then put into water, the nitrate of silver dissolving while the solid sediment can be filtered off. To the silver nitrate solution is added about 3 oz. mercury for each oz. of silver coin, a very small quantity of nitric acid, and a few small pieces of clean and bright iron. After some days, the silver nitrate will have entirely decomposed, and metallic silver will have combined with the mercury to form a pasty amalgam.

Electro-silvering can rarely be satisfactorily carried out in a stamp mill, and it is usual to purchase electro plates from the manufacturers. The silver deposit may be in any desired amount, and varies from 1 to 4 oz. silver per sq. ft., 2 oz. being about the average. All preliminary annealing and polishing having been done by the electro-platers, the amalgamating of these plates becomes a very simple matter, and consists usually in rubbing into the moistened surface of the plate some mercury to which a little sodium amalgam has been added.

Muntz-metal plates need no annealing, but must be cleaned and polished, and then amalgamated in the same way as plain copper, using weak sulphuric acid as a cleanser instead of sal-ammoniac.

*Fixing Plates.*—The amalgamating surface presented to the issuing pulp is often made to commence immediately outside the screen, by cutting a piece of amalgamated copper to fit the lip of the mortar. This is called the "lip" plate, and

is so fastened in position that it can be very readily removed, both for the clean-up and at any time when it may be necessary to open the box.

Of much greater importance, however, is the "apron" plate, extending generally for several yards, and of at least the full width of the box, the use of a spreader and considerably greater width being often desirable, especially where the high duty of the battery produces a stream of great volume. The apron-plate is laid upon a well made and smoothly finished table of wooden planking, carried on supports which may or may not be independent of the battery ground sills. The preference for independent foundations is to be traced to a desire to avoid vibration, which, in the opinion of some, is detrimental to accumulation of amalgam. The supports usually take the form of stout trestles, levelled with the utmost precision, and fixed so that nothing can disturb them; but these are sometimes replaced by an overhead structure and strong depending iron rods. In either case, the table must have substantial wooden sides, not less than 6 in. deep and 2 in. thick. To avoid waste, the plates should be ordered of the exact dimensions of the table, or the table be made to suit the available size of copper sheet, the length of the sheet being the guide to the width of the table. The number of sheets is multiplied till the full length of the table is provided for, their edges merely abutting, or in some cases being brazed together. It is best to obviate the use of any screw, nail, or plug penetrating the plates, and to hold them in place by stout wooden cleats fastened to the sides of the table. The whole table is set with a downward pitch or slope, which must be absolutely uniform and constant throughout. The gradient will depend upon the consistence of the pulp, the size of the particles, and the proportion and character of the sulphurets. It should just suffice to maintain a steady, even and thin stream, the pitch being increased and the volume of water decreased, rather than the contrary. Usually the best grade will be found somewhere between ½ in. and 2 in. to the ft. Its adjustment is a matter of no small moment, the efficiency of the apparatus from a mechanical standpoint being entirely

controlled by it. Without extreme care and precision, there will be a packing of the pulp on some parts of the plate, while other parts will be left uncovered, and over others yet again there will be currents of enhanced volume and speed. Any departure from absolute equality in the stream of pulp is prejudicial to good amalgamating effect. The factor governing the length of table is the fineness of the gold, and the time it therefore needs for deposition.

In Fig. 72 is given a general view of apron-plates.

Suspended tables possess two important advantages over those which are mounted on trestles. With the latter, an alteration of pitch, which may at any time be rendered highly advisable by a change in the ore, more especially in mills doing public work, is a most troublesome and laborious matter, while with the former it can be accomplished to a nicety in a few minutes by merely turning the adjusting screws. To the suspended table also can very easily be given a regular vibratory movement, which has been found very effective in throwing down the valuable particles of the pulp by giving repeated checks to their onward passage. For this purpose, an end shake is to be preferred. Some authorities condemn vibration, and would dispense with lip-plates for that reason.

Sometimes the area of the table is divided into several sections transversely, and stepped. The best way is to make each section accord with the dimensions of the copper plates, so that no joining is necessary. The step should not exceed about 2 in. in depth, or it is liable to cause scouring where the pulp falls. Shallow steps are undoubtedly useful. They admit of a very slight reduction in the gradient, and break up any stratification there may have been in the stream, while it will be found that the splash is instrumental in aiding amalgamation, a deposit forming more rapidly and abundantly around the drip than on any other part of the plate.

An innovation sometimes adopted, partly with a view of saving space, and partly it would seem because the beneficial effect of a splash has been noticed, is the provision of small independent strips of amalgamated sheet metal immediately in front of the mortar. These are called splash-plates. Instead

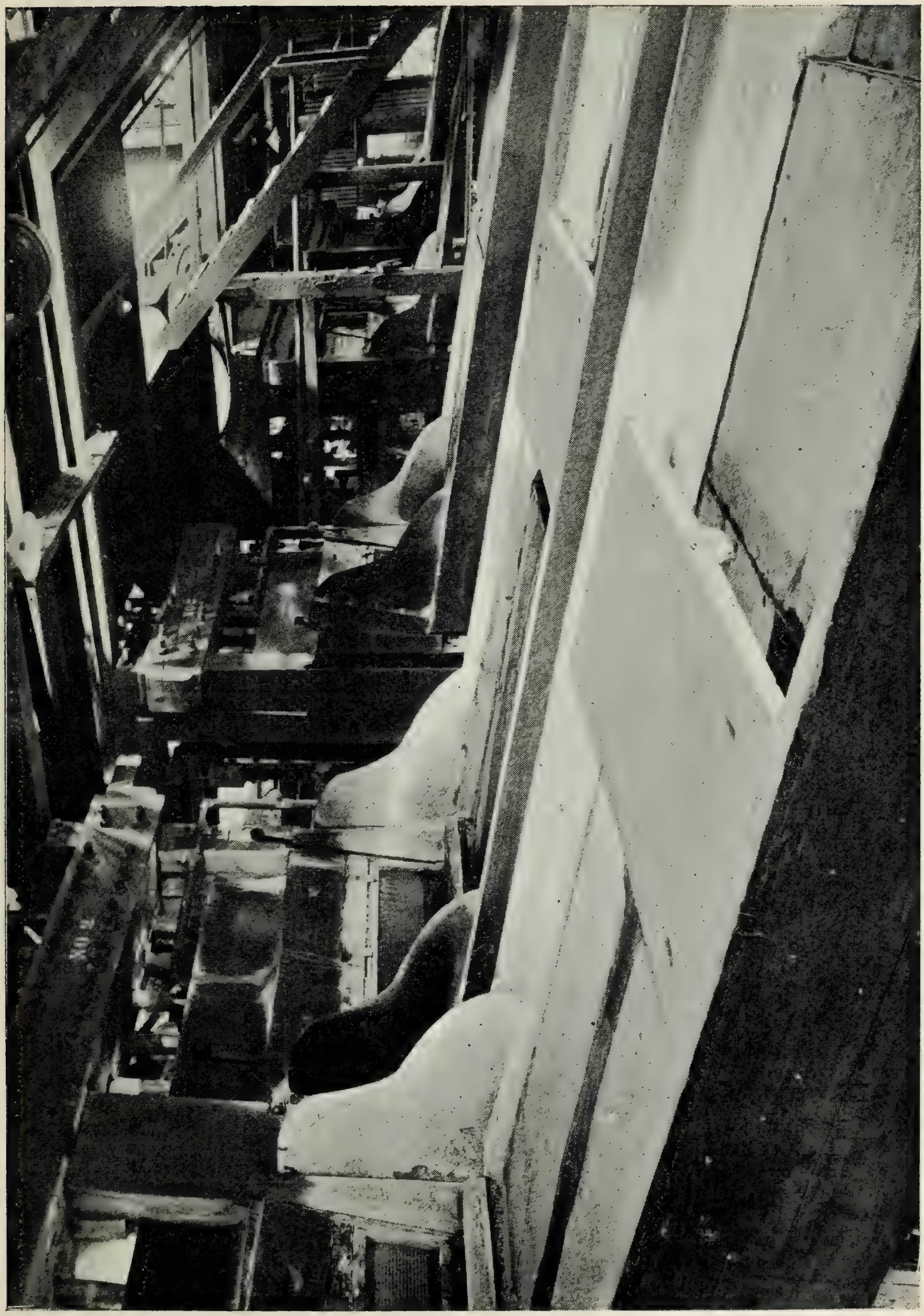


FIG. 72.—APRON-PLATES.

of being laid nearly flat, they are set almost on edge, and occasionally several are superposed, arranged zigzag fashion, alternately sloping towards and away from the screens. Fig. 73 gives an idea of some of these fancy arrangements. In A, *a* is the screen, *b* the battery-plate, *c* the splash-plate, and *d* the apron ; in B, *a* is the screen, *c* the splash-plate, and

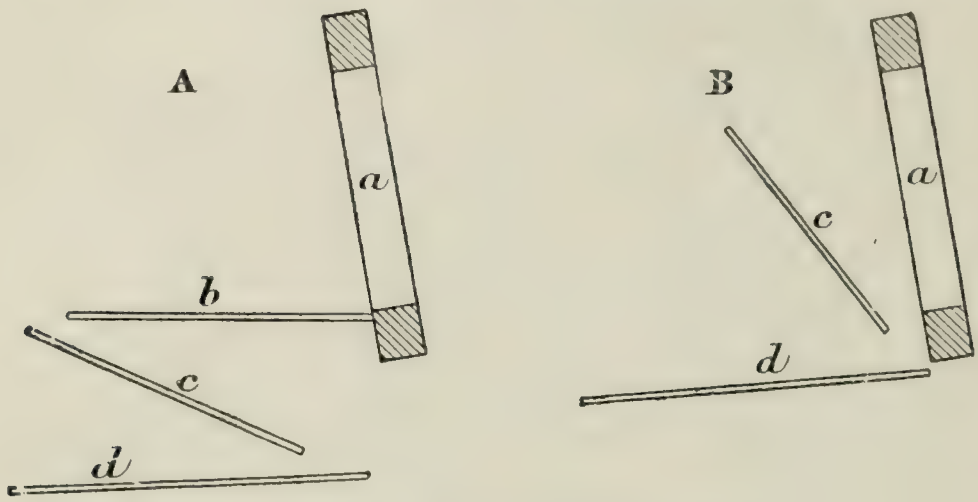



Fig. 73.—Splash-Plates.

*d* the apron. The most useful function of these plates is to get an additional area of amalgamating surface where it is inconvenient to prolong the apron.

*Choice of Plates.* — Plain copper, electro-silvered, and Muntz-metal plates all have their advocates.

The plain copper plate is by far the most generally used, and, when properly prepared and looked after, it is not surpassed in efficiency by any other. It is considered by some authorities to be less active than the electro-silvered plate in catching fine gold. Thus Rickard * quotes a single case where a plain copper plate occupying the third section of the table caught less amalgam than an electro-silvered plate of the same rank on an adjoining table, the ore milled in the two batteries being to all intents and purposes identical. This seems somewhat weak evidence for a general statement. At Lucknow, both plain copper and electro-silvered plates are in use, and the experience there is vastly in favour of the former. In discussing evidence on the influence of temperature of battery water on amalgamation (pp. 215–9), the fact is clearly

* 'Stamp Milling of Gold Ores,' p. 158.

brought out that the plain copper plate is superior, probably from its greater absorptive power.

For the electro-silvered plate, it is argued with some truth that it retains a good surface with less attention than the plain copper, being rather less prone to show verdigris. For this reason, it is preferred where the ore is so poor as not to keep the plates well supplied with amalgam. Yet in the Transvaal, where nearly all the ore is low grade, very few electro plates are to be seen, and no claim is made for their superiority where they are used. At Lucknow, running on ore that did not yield more than 5 dwt. of bullion per ton, their efficiency was much inferior to plain copper. Probably the fact of their requiring less attention to keep bright is a principal reason for partiality to them. They are most popular in California.

At the Drumlummon mill, Montana, R. T. Bayliss * concluded from a series of tests that the silver contents of the bullion showed a persistent tendency to escape amalgamation; the fineness of the amalgam, as measured in gold, being highest nearest to the battery, and giving place to a constantly increasing proportion of silver as the amalgam was deposited at greater distances from the battery discharge.

A. J. Clark † at the Homestake mills, Dakota, confirms this view. His figures are as follows:—

| Sample from | Fineness. | | Ratio of Gold to Silver. |
|---|---|---|---|
| | Gold. | Silver. | |
| Inside plate . . . . . . . | 818 | 168 | 4·87 to 1 |
| Copper plate, 1st row . . . | 812 | 175 | 4·65 to 1 |
| Silvered plate, 2nd row . . . | 654 | 331 | 1·97 to 1 |
| Silvered plate, 3rd row . . . | 618 | 376 | 1·64 to 1 |
| Silvered plate, 4th row . . . | 513 | 465 | 1·10 to 1 |
| Copper plate, head of 1st row . | 809 | 180 | 4·49 to 1 |
| Copper plate, foot of 1st row . | 784 | 185 | 4·14 to 1 |

Difference in average gold-fineness between 2nd and 4th rows, 141.

* Trans. Am. Inst. Min. Engs., xxvi. 33 (1896).

† En. & Min. Jl., Sept. 30, 1899.

In these trials, the silvered plates carried but a light deposit of silver when put down, and, having been some time in use, had probably lost much of it, and almost reverted to the condition of plain copper, as some further tests made later gave the following results:—

| Sample from | Highest Gold-Fineness. | Highest Silver. | Lowest Gold. | Lowest Silver. | Average Gold. | Average Silver. | Average Ratio. |
|---|---|---|---|---|---|---|---|
| Silvered plate, 2nd row . | 573 | 429 | 558 | 408 | 564 | 421 | 1·34 |
| Silvered plate, 3rd row. . | 480 | 433 | 555 | 410 | 569 | 422 | 1·35 |
| Silvered plate, 4th row. . | 566 | 452 | 542 | 425 | 558 | 433 | 1·29 |

Difference in average gold-fineness between 2nd and 4th rows, 6.

Here a heavy silver deposit and shorter service make it likely that the silvered surface was intact, and from this Clark deduces that the explanation must rest upon the amalgamated surface—copper in the one case, and silver in the other; and that the tendency of the silver in the pulp to resist amalgamation decreases, if it does not entirely disappear, when thoroughly silver-plated surfaces are used in amalgamating.

G. E. Collins,* in making working tests at the Reynolds mill, Georgia, to determine the comparative saving on plain copper and electro-silvered plates respectively, obtained the results shown below:—

| Month. | Plate. | Fineness of Bullion. | | Ratio, Gold to Silver. |
|---|---|---|---|---|
| | | Gold. | Silver. | |
| February 1899 . . | Silver-plated | 639 | 151 | 4·23 to 1 |
| | Copper | 619 | 82 | 7·55 to 1 |
| March 1899 . . . | Silver-plated | 798 | 107 | 7·46 to 1 |
| | Copper | 732 | 72 | 10·17 to 1 |
| March 1899 . . . | Silver-plated | 801 | 180 | 4·45 to 1 |
| | Copper | 770 | 207 | 3·72 to 1 |
| April 1899 . . . | Silver-plated | 821 | 162 | 5·07 to 1 |
| | Copper | 867 | 127 | 6 82 to 1 |

* En. & Min. Jl., Dec. 23, 1899.

Despite great variations, due probably to changes in the ore, the nett result is a much higher ratio of silver from the electro plate than from the copper. But there was a marked loss of silver from the plate. After tests yielding 170 oz. bullion, with an average silver assay of ·041 oz. higher than that from the plain copper, the total excess of silver only amounted to 6·97 oz. Collins very pertinently asks whether this may not have been derived entirely from the plate, seeing that it is only equivalent to 18·7 % of its original charge (1 oz. per ft. for 37·33 sq. ft.); and he questions whether the heavy silver-plating—2 or 3 oz. per sq. ft.—sometimes adopted in gold-mills, is not a mistake, and whether it would not be more economical to use a thin coating—1 oz. or less—renewing it, if necessary, as the silver wears off.

Some data collected by E. A. H. Tays * at San Jose de Gracia, Mexico, while verifying the increase in silver contents of the bullion in proportion to the distance from the screen at which it is deposited, lead to the inference that, if the whole of the bullion be collected together and assayed, the ratio of gold to silver will be found to accord very closely with that constituting the average of the ore.

Certainly this was so in the author's experience at Lucknow, the variations being very slight; and it seems only reasonable to suppose that the bullion of highest standard, having the greatest specific gravity, should be deposited first. After all, gravity is far and away the leading factor in amalgamation, whether inside or outside, and differences in specific gravity seem quite as rational an explanation as to suppose a "selective" action by certain plates. With very low standard bullion, the phenomenon will probably be most marked, as its lightness will exaggerate the distance travelled before settlement.

The Muntz-metal plate came into notice years ago on the Thames goldfield, New Zealand, through a temporary local scarcity of copper plates, and has been adhered to in some measure ever since. It is much cheaper, the alloy itself being lower priced than copper, and in addition considerably less

* En. & Min. Jl, Feb. 24, 1900.

metal needs to be used, because, its action being quite superficial, there is no absorption of mercury or amalgam, and scraping and steaming can be dispensed with, thus prolonging the endurance of the plate. Owing to its non-retentive character, the clean-up is much simplified, but must be repeated at greatly shortened intervals. It is claimed that the two metals composing the alloy, copper and zinc, forming a galvanic couple, set up a mild electrolytic action on the water flowing with the pulp, whereby hydrogen is evolved. This, being in the nascent state, is exceedingly energetic in "reducing" those sulphides which cause the sickening of mercury and consequent patches of verdigris on the plates. Therefore the Muntz-metal plate is extolled by Rickard,* when the ore treated is of such a character.

But it is a little difficult to understand why it should be condemned by the same authority for material containing sulphuric acid or sulphates arising from the oxidation of pyrites, and preference then be given to copper—"a heavy scum is formed upon the Muntz, while the acid tends to keep the copper clean." The matter requires some elucidation. One would suppose an acidulation of the water at any rate to be necessary for giving an existence to the electrolytic action, without which the reducing hydrogen cannot be generated.

Thus Vautin † says Muntz metal "can be kept in a perfect condition by a weak solution of sulphuric acid." He narrates that in certain experiments he "prepared copper plates for amalgamation by means of nitric acid, cyanide of potassium, and zinc amalgam, and suspended the plates so prepared in a weak solution of sulphide of iron and mine waters. The nitric acid prepared plate was the first to become coated, cyanide next, and the zinc amalgamated last, even that not remaining bright for more than an hour; but when a well prepared copper plate is in contact with metallic iron or zinc, as in a galvanic couple, the mercury is in that case very little affected by the acidity of the water."

Whilst its advocates show such a wide divergence of

* 'Stamp Milling of Gold Ores,' p. 181.

† Trans. Inst. M. & M., i. 228 (1893).

opinion on its vital qualities, Muntz metal is not likely to have extended application. But Rickard further regards it as "particularly suited to custom mills," because it permits a complete and exhaustive clean-up. That certainly is in favour of the customer, but will hardly appeal to the owner of the custom mill. The customer is not allowed to scrape or steam the copper plates, so that their absorptions are a direct gain to the owner of the mill. Muntz metal effectually disposes of this handsome "perquisite."

*By Wells.*—A well, trough, or riffle, supplied with mercury, is an essential feature in Australian batteries, as an

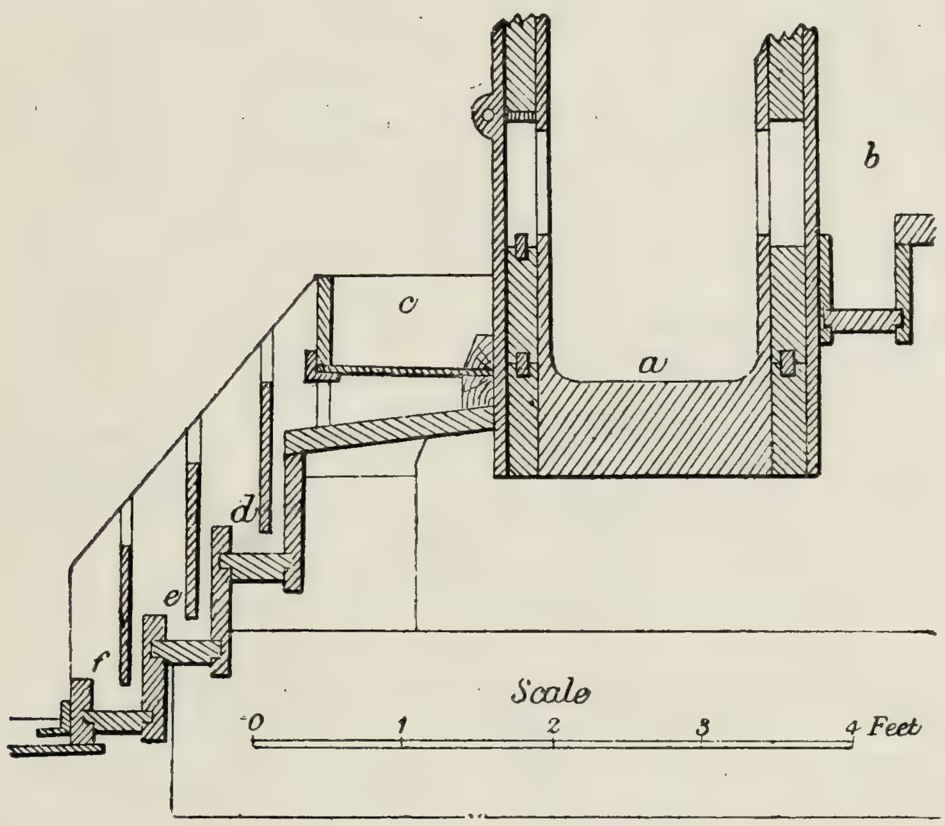



FIG. 74.—MERCURY TROUGHS.

adjunct to amalgamated plates, and in a very few instances even replaces them. The details of the arrangement vary considerably, but quite a common type is illustrated in Fig. 74. The material issuing from the mortar *a*, both by the ordinary front discharge and by the back discharge *b*, when that is adopted, flows into the distributing trough *c*, which is furnished with a perforated plate in the bottom, to arrest any coarse stuff that may have escaped from the box. From the distributor, the pulp passes to three drop-wells *d e f* in succession, falling in the first case about 10 in., in the second about 9 in., and in the third about 8 in. Each of these wells or troughs is fitted

with a splash-board, which is just immersed in the mercury bath, so that all matters entering the trough are compelled to traverse the mercury before they can escape over the lip. The troughs are set with a very slight inclination towards one end, where a tap hole is provided for drawing off the charge of mercury and its accumulated amalgam. The whole contrivance is covered in securely, and locked up to prevent stealing.

Another style is shown in Fig. 75 (scale $\frac{3}{8}$ in. = 1 ft.). In addition to the mercury wells proper, marked *a*, and having the usual splash-board *b*, smaller transverse recesses called riffles are introduced, one being just below the upper well and the other just above the lower one. These riffles are not charged with mercury.

In the two examples given, no plates are employed; but

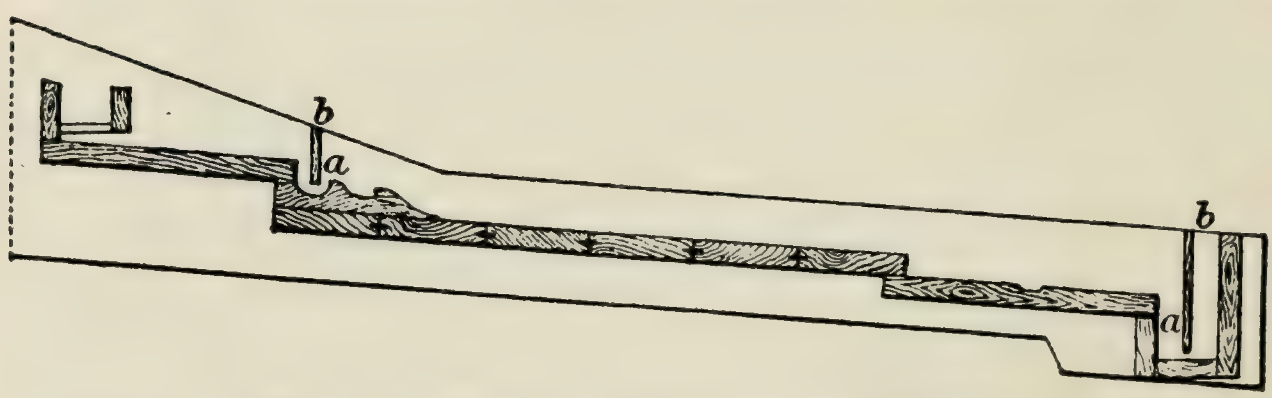



FIG. 75.—MERCURY TROUGHS AND RIFFLES.

in many other cases, wells or riffles are interposed between the various sections of the plates. In these instances, the wells are made of iron, which is preferable to the ordinary wooden structure, the iron serving to assist in keeping the mercury bright.

In the Otago district of New Zealand, where low winter temperatures are encountered, use is sometimes made of a steam-jacketed riffle, as in Fig. 76. A pipe *a* conducts live steam into the jackets or double bottoms *e* of the riffles *b c*, the supply being regulated by cocks *d*, so that the degree of heating is under control, other cocks, not shown, emitting the condensed water. Amalgamated copper plates are placed at *f*, and a flat mercury trough at *g*.

The potency of mercury wells and riffles as amalgamators has been much exaggerated. In their existing forms, it is a

universal experience that, of the total amalgam caught in a mill using plates and wells, the proportion saved by the latter is absurdly small, and for the following reasons. In attempting to pass pulp through a bath of mercury, it will be found that instead of the desired dissemination of the particles evenly through the mercury, there will be a distinct tendency to consolidate into certain channels, a mass of pulp being necessary to overcome the cohesion of the mercury and force a passage. Individual particles are quite unable to exert this power, and can only do so when their number amounts to a considerable

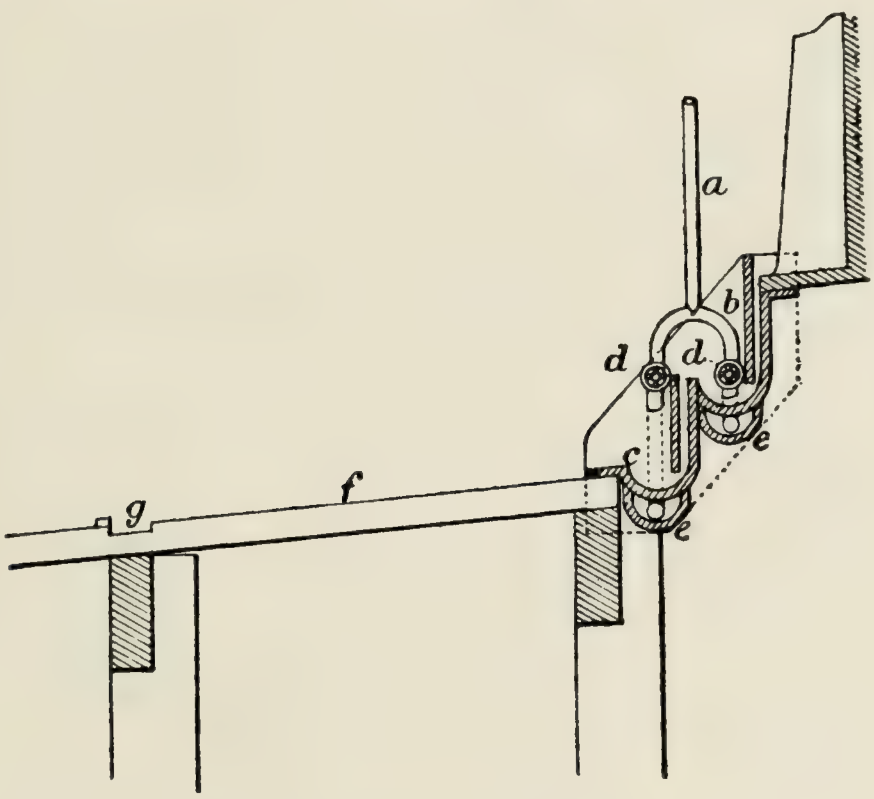



FIG. 76.—STEAM RIFFLES.

volume or mass. The intimate contact between gold speck and mercury surface which is the *sine quâ non* of amalgamation is thus a fiction, so far as wells and riffles are concerned, and nothing but coarse metal, which can rely upon its specific gravity alone, is caught in them. When the drop is deep, even relatively coarse grains may escape, owing to the excessive agitation. The great majority of milling ores carry considerable pyrites, ranging from 1 to 3 %, and a portion of the sulphurets is often in comparatively coarse grains—very coarse relatively to the gold particles. These by their weight and mass do get caught in the wells and riffles, loading the surface of the mercury so as to present a mechanical obstacle to amalgamation,

and often contributing a chemical impediment by sickening the mercury. To some extent, this latter objection may be simply remedied by the methods already described, the mercury bath lending itself well to galvanic treatment. But the mechanical drawbacks seem insurmountable. The effective area presented is very limited, and if wells were to be multiplied so as to approximate the space covered by plates, the quantity and cost of mercury would be extravagantly increased. The legitimate use of wells is as traps for arresting excess mercury which may have rolled off the plates, and detached particles of amalgam which get away with them. For this purpose, it would seem that a shallow (about 3-in.) gutter is better adapted than a deep trap, in which there must necessarily be great swirl, affording but little opportunity for the escaping particles to come to rest. If one such shallow trap is not sufficient, the number should be augmented rather than the size.

*The Clean-up.* — Periodically, the frequency depending partly on the richness of the ore being milled, a "clean-up" is made, each battery being taken in turn. The stamps are hung up and the screen is removed, so as to give access to the interior of the box. The dies are lifted out of place by a chisel bar, and washed in a bucket or tub of water, any adhering amalgam being carefully detached. All material, crushed and uncrushed, lying in the box is collected in buckets and screened through about $\frac{1}{4}$-in. sieve into a tub. The coarse is picked over for removal of any large pieces of steel, such as bolt-heads, drill-bits, and chips off shoes, etc., and returned to a feeder. The fines are carried to the clean-up trough, which is provided at one or both ends of the battery, fitted with convenient shelves and racks, and with water and steam pipes, the overflow from the trough leading to the tailings launder. In some large mills a track is provided for carrying a trolley, so that transportation of amalgam, and of dies, plates, etc., to the clean-up room is facilitated. At the trough, the fines are panned off, the rejections being filled into buckets and carried or trollied back to the box for use in re-packing the dies. The heavy matters remaining in the dish are

repeatedly stirred with a powerful magnet to eliminate the thousands of tiny particles of iron. These last are spread in trays exposed to the weather to undergo slow oxidation, and, after months of exposure, a small quantity of amalgam may be panned from them. The iron-free headings from the dish are subjected, together with other amalgam, to a grinding with additional mercury in a clean-up pan of the Berdan or other type.

The inside plates are rubbed or scraped *in situ* if conveniently placed, or removed and treated in similar fashion, often being immersed in hot water or submitted to a steam hose, as a preliminary, to soften the amalgam and facilitate its detachment. Chuck-block and screen-frame plates are always removed for cleaning up. Apron-plates are dealt with as they lie, square rubber blocks, steel scrapers and chisels being used according as the amalgam is easy or difficult of removal. Hard stubby whisk brooms and stiff-haired scrubbing brushes are also employed. The cleaning proceeds from the top downwards, every inch of surface being carefully gone over. Large palette knives or spatulas, 8 to 12 by 1 in. in the blade, are useful for picking up the collected amalgam and conveying it to enamelled-iron basins or buckets, which need only be of very modest dimensions, as the weight of amalgam is such as to render a large vessel unhandy. Good plates absorb quite a large proportion of amalgam, and hold it so tenaciously that the mercury must be liquefied by heat before it can be availed of. Some authorities recommend this heat to be applied by heaping hot sand upon the plates, but a steam-jet is so much more efficient, rapid, and easy of management, that its services should always be provided for in preference. Usually steaming or sweating the plates is only done at longer intervals, say once or twice a year, while the ordinary clean-up is generally monthly, or in accordance with the pay-days, so that the revenue from it may provide the money for the wages and store sheets.

The contents of mercury wells is run off and strained through canvas, calico, or shammy leather, the liquid mercury being returned to them after cleaning (see p. 209), while the

solids extracted are treated in the clean-up pan. The squeezing of amalgam by hand is a tedious process and very tiring, so that when large quantities have to be dealt with, some mechanical contrivance is advantageous. A hydraulic squeezer is made by Fraser & Chalmers. The amalgam, in a canvas bag, is placed in the perforated cylinder, and the ram is forced against it. To liberate the dry amalgam, the valve is reversed, and the ram is thereby withdrawn. A centrifugal separator has also been introduced, but is not in general use. The drier the amalgam, the less retorting is needed afterwards. No mechanical appliance yet introduced gives as dry an amalgam as is got by hand squeezing.

While plates are in use as amalgamators, they should never be scraped or sweated to such a degree as to expose the copper, or their future activity will be much impaired. No amount of steaming and scraping, however, can separate all the gold they have absorbed, and apparently about $\frac{1}{4}$ oz. gold bullion per sq. ft. of copper surface is so intimately combined, that it remains even when the copper is melted down. Old plate scrap thus has a high value.

After every clean-up, and indeed whenever there has been any impairment, mechanical or chemical, of their bright "quick" surface, plates must be "dressed." This dressing consists in brushing them thoroughly with a stumpy whisk broom or a hard scrubbing brush, adding a sprinkling of weak cyanide liquor (p. 227) if plain copper, or of very dilute sulphuric acid if electro-silvered, and then rubbing in fresh mercury.

*Impure Amalgam.*—It is rarely safe to assume that amalgam will consist of nothing but gold and mercury. While it is asserted that base metals, such as iron and antimony, will not amalgamate under "ordinary" conditions, there frequently arise such conditions, whether ordinary or extraordinary, as result in a most bulky amalgam containing much of these metals.

A. J. Bensusan * records a quantity of gold-iron amalgam taken chiefly from the inside plates of a 5-stamp mill crushing

* En. & Min. Jl., June 25, 1898.

felsite schist containing a fraction of an ounce per ton. The phenomenon only appeared occasionally, when a scum would form unexpectedly, and the amalgam, both inside and outside the boxes and on the plates, would become a dark grey. After retorting, analysis showed it to contain 50 % metallic iron. The battery boxes, stamp shoes, etc., became coated with this iron-gold amalgam.

In another case, at Peak Hill, New South Wales, from E. M. Mardin's experience, sometimes a slimy amalgam was produced, which, when retorted, gave a black mass, consisting chiefly of copper oxide, iron and gold. Its source was the pyritous ore, containing a small proportion of copper, which was roughly roasted in heaps. Doubtless a portion of the copper was converted to sulphate only, and the battery water being saved and used repeatedly, gradually copper salts accumulated in solution, acting on screens, and on fragments worn from shoes, dies, etc., and forming a copper amalgam which would envelop considerable iron.

Similarly, J. C. F. Johnson * encountered amalgam from a West Australian mine (presumably recovered from battery and plates) containing nearly 70 % of impurities, principally iron, derived in this case "from arsenite of iron in the ore, and large quantities of calcium, sodium, and magnesium chlorides in the mine water used for milling."

Iron readily amalgamates in presence of sodium amalgam, and, on cleaning up batteries where sodium amalgam is used, the iron is frequently noticed to be slightly amalgamated in spots. In experiments with the Designolles process in the United States, difficulty was experienced through the formation of iron amalgam ; and amalgamation of iron has been noticed often in pan amalgamation, where copper and salt are employed.

The remedy, after retorting to recover as much of the mercury as possible, is to subject the residue to a thorough oxidising roast, and then wash out the ferric oxide by means of sulphuric acid, before proceeding to melt the bullion.

When native antimony occurs in the ore, this forms a most

* En & Min. Jl., June 25, 1898.

voluminous amalgam, and while offering no hindrance whatever to amalgamation of the gold, it gives considerable trouble in the clean-up. By thorough grinding and washing in the clean-up pan, the bulk of the antimonial amalgam (carrying about 10 % gold) can be separated from the gold amalgam pretty thoroughly, though the latter, and even the melted bullion from it, will always contain notable traces of antimony. At Lucknow, N.S.W., in the author's time, it was a common occurrence to have 100 to 200 oz. antimonial amalgam, or "dross," as it was locally termed, from a monthly clean-up, and at an earlier date in the history of the mine, when very rich ore was being milled, the accumulation has been such as to actually choke the battery. The practice was to separate this dross as completely as possible from the gold amalgam, squeeze it to the utmost, and then expose it on iron trays in the open air to a steady roasting, thereby removing the antimony by volatilisation and oxidation, and of course losing such mercury as was associated with it. The roasted residue was reground with clean mercury. The site chosen for the roasting was where persons could not be affected by the noxious fumes emitted. The latter are so abundant that the operation is not adapted to the retorting room, unless most efficient ventilation is provided.

*Current Practice.*—A few examples of current practice in outside amalgamation cannot fail to be interesting.

Among Californian mills, at the North Star the pulp passing from the batteries drops 7 in. on to a lip-plate 52 in. wide and 18 in. long, sloping $1\frac{1}{2}$ in. per ft. Across the lower end of the lip-plate is a distributing trough 52 in. long × 3 in. × 3 in., with $\frac{3}{4}$-in. holes in the front side. From this, the pulp falls $2\frac{1}{2}$ in. to the apron-plate, 56 in. wide and 48 in. long; it then traverses a sluice-plate 48 in. wide and 15 ft. 9 in. long, and a trough 3 in. wide and 48 in. long, and finally passes the mercury traps (Robert).

The sluice-plate is a feature of the Californian battery, and generally does not exceed 24 in. in width, often being much less. The principle on which it is supposed that pulp which fails to deposit its gold on the abundant area of the

apron-plate will be induced to do so on a sluice-plate of less than half the width, and consequently more than double the current, is a conundrum which awaits solution : a wooden or iron launder would be just as serviceable and much less costly. Obviously, to secure increased deposition, the area of the plate must be increased instead of diminished. The outside plates are scraped and dressed every morning. They are all electro-silvered (1 oz. per sq. ft.), and inclined 1½ in. per ft.

At the Empire mill the plates carry 2½ oz. silver per sq. ft. ; and at the W. Y. O. D., 5 oz., the apron-plate at the latter being 14 ft. long and 50 in. wide.

The Gover battery has an elaborate arrangement. The issuing pulp falls first on a "battery" plate, 50½ in. by 19 in., set almost horizontal, and discharging on to the upper edge of a "splash" plate, 46 in. by 8 in., set at an angle of 45° ; thence it flows over the apron-plate 50 in. by 36 in., set at 1¼ in. fall per ft., and reaches a small trough or riffle. This has two apertures, delivering into distributors, which are of copper plate with perforations 1 in. apart ; these are to break up the stream and distribute it evenly over the two sluice-plates, 11 ft. by 14½ in., set at 1½ in. per ft., which again discharge on to a single sluice-plate 6 ft. by 14 in. As 90 % of the amalgam won is got inside the mortar, outside plates are a secondary consideration ; how any useful duty can be done by the sluice-plates is a mystery.

A novelty at the Utica mill is a 12 in. by 1 in. rough board instead of an ordinary amalgamated copper lip-plate. It is said * that, after a month's use, it is an efficient gold saver, and can be cleaned in a fraction of the time required by a plate.

The batteries of Gilpin County, Colorado, mostly have about 12 ft. of 48-in. plate, plain copper, set at 1½ to 2⅛ in. per ft. They are cleaned up once in 24 hours, and dressed every 12 hours with weak (1 oz. in 1½ gal.) cyanide solution.

In Dakota, the batteries of the Homestake Co. have splash-plates, set so as to minimise the scouring effects arising from the aprons being placed so far below the screens that the pulp

* 'Mineral Industry,' vii. 351 (1898).

must fall 6 to 10 in. At the Golden Star mill, which is a good example, the first plate of the apron is of plain copper, 54 in. by 12 ft.; the second, third, and fourth "rows" are electro-silvered (1 oz. per sq. ft.), and of the same size. Riffles about 2 in. deep and traps 18 in. deep are provided for arresting escaped mercury; the former do much the better work. The plates are dressed occasionally "with a weak solution of salt and sulphuric acid—a handful of salt and ½ teacupful of acid in 2½ gal. water." (Rickard.)

At Mesquital, Mexico, the tables are 11 ft. 9 in. long by 54 in. wide, the inclination being 1¼ in. per ft. Originally they were 16 ft. long, but were shortened, as practically no amalgam was deposited below 8 ft. from the screens. (Claudet.)

The Lucknow, N.S.W., battery is provided with lip-plates and three successive rows of apron-plate, followed by the ridiculous Californian sluice-plate, the mill having been erected by a Californian. The great majority of the plates are plain copper, but a few electro plates have been tried in various positions. Invariably better efficiency is obtained from plain copper than from electro-silver. Drops between the several rows of plate have given good results, when not so deep as to cause a scour. The sluice-plates are mere launders, and catch nothing.

Typical mills on the Thames, New Zealand, have short plates (7 or 8 ft.) in three steps, with intervening riffles about 2 in. deep, some containing mercury, others none. The plates are mostly of Muntz metal.

In the Transvaal, while there are almost endless variations in the size, arrangement, and number of the plates, plain copper is nearly universal. Apron-plates are in constant favour, their width being generally about 54 in., length 10 to 12 ft., and pitch 1¼ to 1½ in. per ft.; they are mostly in one unbroken table. The adoption of lip- and splash-plates is quite exceptional, and the same may be said of riffles or troughs, but a mercury trap is usual. But little more than the first 2 ft. of the apron catches any gold, and while this portion is cleaned up daily, the remainder does not demand attention

oftener than once or twice a week. The only dressing used is dilute potassium cyanide solution. The Primrose mill has lip-plates 1 ft. wide sloping about 2 in. per ft., and splash-plates 14 in. wide slanting towards the screens, as well as riffles, but the latter are useless. At the Jumpers mill, the outside plates have been lengthened from 9 ft. to 12½ ft., and inside amalgamation has been dispensed with ; the result is said to be very satisfactory.

In the Urals, some plates described by D. A. Louis are set at an incline of 1 in 7, in series of 4 plates with a drop of 2 in. after each, the plates being 30 in. wide and 5 ft. long, except the uppermost, which is only 3 ft. long. Some tables have 11 plates, each 11 in. wide only, and stepped 1½ in.

Victorian practice differs widely, but the mercury well is an almost invariable adjunct, and in some cases is used alone. With the heavy free gold at Clunes, 80 % is arrested in and immediately outside the box, without any plates. Skimming removes supernatant mineral, and the weekly clean-up consists in squeezing the excess mercury out of the amalgam and returning it to the wells. A typical Bendigo mill recovers about 20 % of its amalgam on the apron-plates and 7 % in the wells ; the former are 11 ft. long, 60 in. wide, slope 1¾ in. per ft., and demand only a fortnightly clean-up. But in another representative Bendigo battery, the outside catchment of about 40 % is nearly evenly divided between apron and well ; and again in another, 8 % only comes from the wells and no less than 58 % from the apron-plates, which have the exceptionally low grade of ¾ in. per ft., rendered possible by an excess of water demanded by the double discharge of the battery.

**Amalgamating Machines.**—In a sense, the ordinary stamp battery falls under this heading, and in very many cases it is an amalgamating machine, and a highly efficient one ; but its true and original office is crushing, and the amalgamation process carried on within it is incidental, one may say. On the other hand, a number of machines have been invented with the primary object of carrying amalgama-

tion to the utmost possible degree of perfection ; and while it must be admitted that the majority have signally failed to attain an ideal efficiency, and that recent advances in leaching processes have robbed amalgamation methods of much of their importance, there are a few typical machines which cannot be altogether passed by. One principle underlies them all, viz. to bring about a more intimate commingling of the mercury and gold particles than can be ensured by reliance on the specific gravity of the latter, and this is sought to be accomplished by rubbing or grinding the mercury and pulp together.

*Arrastras.* — The arrastra is one of the oldest of these machines, if it is not the actual prototype of them all. In universal use for many generations among the Spanish silver miners of South America and Mexico, it was copied by the early Californian gold miners, and has maintained a recognised position in metallurgical operations ever since. Properly an amalgamator simply, it has come to be regarded in some quarters * as a rival to the stamp battery, for small mines, or in the opening stages of new mines, before the time has arrived for putting down expensive machinery. It is a cheap and simple device, adapted to working soft surface ores and extracting practically their full value, while the nature and extent of the deposit are in course of investigation. In most cases, it can be constructed from material which can be obtained on the spot, or without great difficulty ; it is a very close saver of gold ; it can be operated by animal power if no other be at hand ; and it involves only a very small expenditure for construction and operation. True, it is somewhat slow in working, but where the quantities to be handled are comparatively small, this is only a minor defect, and is fully compensated for by its other advantages. Its simplest and perhaps best form consists of a shallow circular bed, paved with very closely set blocks of hard stone. A post is set in the centre, usually upon a raised block, and is supported at the top by a rough timber frame, in such a way that it is free to revolve. This post carries one or more arms, to which the

* En. & Min. Jl., Dec. 23, 1899.

drags are connected by chains. One of the arms extends beyond the outer circle of the bed, and to this the motive power can be attached. This power is most usually, in the South-west States of America, a mule or burro; in Mexico and Central America, one or more peons or Indians are often substituted for the four-legged engine. Fig. 77 shows a sectional view of a Mexican arrastra as usually constructed: A, upright post; B, arms carrying drags or mullers C; D, central block of hard wood supporting the lower end of the post.

The stone bed of the arrastra is usually laid upon a foun-

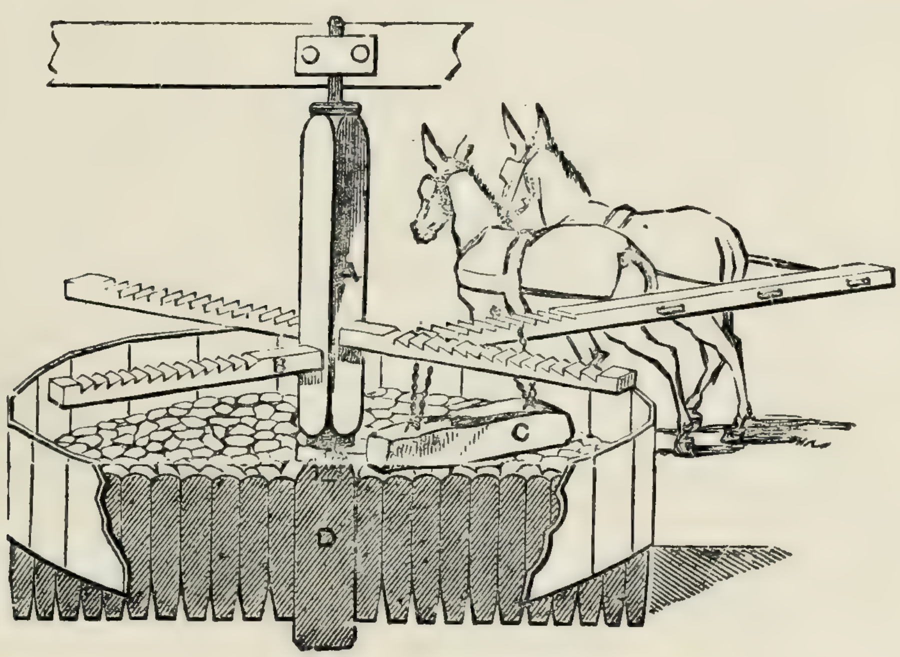



FIG. 77.—ARRASTRA.

dation composed of a layer of clay or soil beaten down as hard as possible; where it can be had, concrete may be advantageously used for the bed. It should project for 2 ft. or so outside of the arrastra floor all round, in order to prevent any mercury which may escape from soaking into the soft earth. Around the bed there must be a curb or wall to prevent the pulp from splashing out; this may be of stone or of wood, as is most convenient. Sometimes the bed of the arrastra is sunk below the surface, and in that case the curb may be simply earth rammed hard. Probably the most satisfactory curb is made of wooden staves. The height will vary from 18 in. to 4 ft., or perhaps more, according to the size, and the quantity which it is proposed to treat. The bed should

be made of blocks of hard stone fitted together as carefully as possible, and evenly laid. The larger the stones the better, as there will be fewer joints to make. The joints should be filled in with cement where it can be had, or with good hard sand well rammed. The choice of stone will in most cases be determined by the neighbourhood. Granite and quartzite are generally considered good. If possible, it is better to have a stone that will not wear smooth by friction, but will retain its roughness of surface, as the efficiency of the grinding is less when the stones are worn smooth.

In the more elaborate machines, the central post is shod at the bottom with iron, and carries an iron pin. A cast-iron shoe may be used as a support, and an iron bearing also at the top. While these are to be preferred, they are not indispensable, and good work has been done with wooden shoes and bearings, which can be worked out very fairly with ordinary tools. The number of arms carried by the post will depend upon the size of the arrastra and the power used. Where a mule is the operator, a plain arm, to the end of which he can be attached, is all that is required. In more elaborate constructions, the central post may carry a bevel wheel at the top, which will gear with a pinion carried on a shaft driven either by a water-wheel or a steam-engine.

The drag-stones or grinders are attached to the arms which project from the post. They are usually stones, as large and heavy as can be procured, and are of the same material as the bed-stone. The face must be flat, and the stone itself should be nearly flat also, and a little wider on the face than on the back, so as to avoid any tendency to turn over while running. Two eye-bolts are secured in the top of the drag-stones, either leaded in, or screwed into wooden plugs driven into holes made in the stone. Chains or raw-hide ropes extend from these eye-bolts to the arms, and should be so arranged that their length can be varied as found necessary. The general custom is to make one of the ropes or chains a little longer than the other, so that the front of the drag-stone forms an angle with a radius drawn from the centre of the arrastra. The object of this is to secure a better turning over and mixing of

the pulp. The stone is so hung that the front end is raised a little from the bottom ; if it rested flat, it would have a tendency to push aside the particles of ore, instead of drawing them underneath and grinding them. The number of drag-stones varies : in a small machine, only 2 may be used, and sometimes the number is as high as 8 ; but 4 is probably the most usual number. A drag-stone 200 lb. in weight is about the smallest that should be used, and stones of 600 or 700 lb. each are not uncommon. The diameter of the arrastra fluctuates : 8 ft. is about the smallest, while 12 ft. is quite common, and they are sometimes built up to 20 ft. diam. For a 12-ft. arrastra, the width of the pavement or grinding bed would be 4 ft. or a little over. The number of revolutions where mule power is used is about 6 per minute ; where water or steam power is available, they may be run to about 12 a minute.

After an arrastra has been built, it should be run for several days on barren quartz or sand until the running is comparatively smooth, and the cracks in the floor have been well filled up. After it is ready for use, the ore is charged in, the quantity varying with the size of the machine, the hardness of the ore, and the fineness to which it has already been broken. Where a rock-breaker is available, it will, of course, be used ; but where the stone must be broken up by hammer, it will naturally be turned over to the machine in larger pieces. In Mexico, the ore for the arrastra is usually picked over very carefully by hand before breaking, in order that as little barren rock as possible may be charged. A usual charge is broken ore sufficient to form a bed $1\frac{1}{2}$ or 2 in. deep all over the floor. This is spread uniformly over the bed, damped down with water, and the drags are started. After the ore has been ground fairly fine, water enough is added to make it rather a stiff paste. When it has been well mixed, mercury is thrown in, generally sprinkled over the surface of the pulp as uniformly as possible. One accepted way of doing this is to squeeze the mercury through a piece of fine canvas, breaking it up into globules. The quantity used is generally about 3 times that of the gold or silver supposed to be in the ore. It is a very com-

mon practice to put it in gradually in small quantities, taking out samples of the pulp from time to time, and panning them off in order to ascertain whether any amalgam is forming. The experienced miner will readily see from the result of this panning whether it is best to add any more mercury. The consistence of the pulp is an important point; it should be thin enough to allow the drags to work through it easily and smoothly, but not so thin that the mercury will sink to the bottom and collect in the depressions of the bed, where it would do little or no good.

The arrastra is a fine grinder and amalgamator, and does its work well if allowed sufficient time, but it cannot be pushed or hurried. For this reason it is not adapted for large plants where great quantities of ore are to be worked, nor is it suitable for very low grade ores, as its operations are too slow to make such ores yield a profit.

When the amalgamation is considered complete, the pulp is drawn off. Arrangements are usually made for this by providing gates or openings in the curb; generally there are several at different levels, all discharging into a sluice-box. Water is run in to thin the pulp and wash out the ground ore. As a rule, most of the amalgam remains on the bottom of the arrastra; but if the gold is fine, the sluice-box should be provided with amalgamated copper plates, or with riffles. Where there are sulphurets in the ore which it is desirable to save, blankets may be used. When the pulp has been drawn off, a new charge is introduced, and the operation is begun again. The time required for working one charge varies according to the nature of the ore and other considerations; probably 6 to 12 hours will be the limits, and the quantity of ore worked will also vary from the same causes.

A partial clean-up is made every week or two, when the arrastra is cleaned out, its contents being removed as closely as possible by scoops and stiff brushes; no attempt, however, is made to scrape out amalgam which may have worked its way down into the cracks, this being left for a general clean-up. The material collected is washed out in pans or a cradle in the usual way, to separate the amalgam, and also to

recover any sulphurets or other material which it may be thought best to save.

A general clean-up is usually made only when the stones of the bed or floor are so far worn out that they have to be replaced. When this is done, the stones are taken up, carefully scraped, and cleaned to save any particles of amalgam which may adhere to them. The sand on which the stones were laid is carefully panned out, and frequently the hard earth or clay bed is scraped off, and the upper part is washed. A new pavement is then put down, if work is to be continued. The length of time which a pavement will last is very variable, as may be supposed. It depends upon the nature of the stone of which the bed and the drags are made, the nature of the ore ground, and some minor considerations. It may happen sometimes that, where a prospector has been engaged in the work, the machine may be abandoned altogether. In that case, there is at least the satisfaction of knowing that no very heavy expense has been incurred in setting it up. Time and labour have been lost, of course, but no considerable amount of money.

Many attempts have been made to improve upon the arrastra, to increase its speed, and to make it a more modern machine in appearance. If old miners are to be believed, these attempts to modernise the machine have not been improvements, and more efficient work can be done with the old-style stone bed than with any of the so-called improvements.

In Spanish America, arrastras are often used for treating concentrates. Thus F. Owen describes * them as dealing with the "heads" from buddles at La Salada. These are 9 ft. diam. and 2 ft. deep, and driven by belting from a water-wheel. Their sides and floors, as well as the 2 drags, are of the hardest granite obtainable. One arrastra of the above dimensions, making 40 rev. a minute, grinds 5 cwt. of pyrites to 100 mesh in 24 hours. But this speed is very rarely attained in native mills. About 25 lb. of finely ground pyrites is taken out of the arrastra during the course of

* Trans. Inst. M. & M., iv. 16 (1896).

each hour, and a similar weight of unground pyrites is put in to replace it.

*Barrels.*—On many fields, the amalgamating barrel survives as a means of separating the free gold, amalgam, and mercury entangled among the finer portion of the sand taken out of the mortar box at clean-ups, and from "blanket sands" when blankets are used as rough concentrators. Such a barrel is shown in Fig. 78. It is constructed of wood or iron, revolving on a pivot at each end, and actuated by toothed

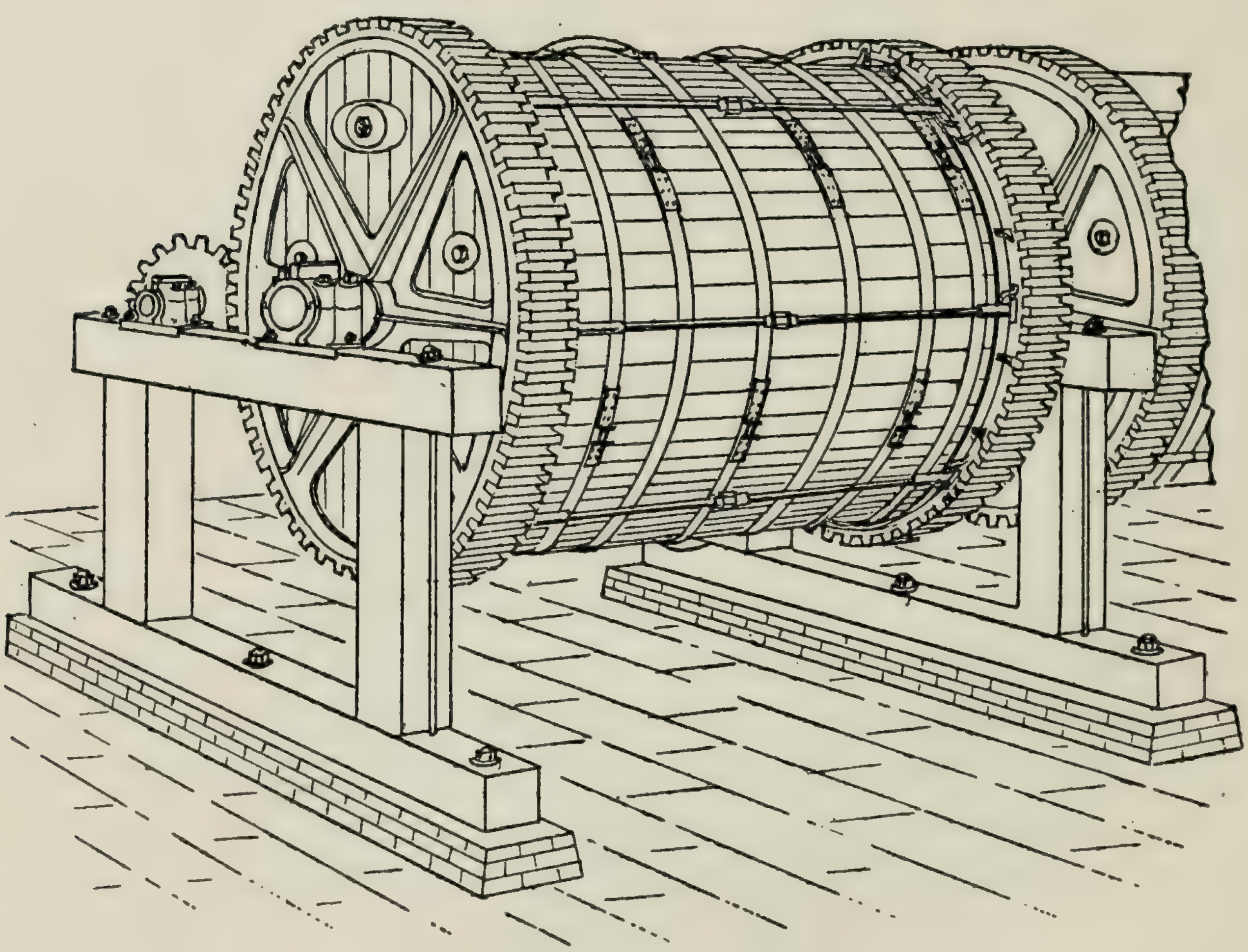


FIG. 78 —AMALGAMATING BARREL.

gear or belting. The size varies considerably, and the weight of the charge follows suit. Sometimes additional mercury is introduced in some quantity, while in other cases that contained in the pulp is deemed sufficient. Water is always added ; cold and hot each has its advocates. The former is perhaps preferable when mineral impurities present would dissolve more readily in hot water, and become more active agents in sickening the mercury. Thorough flouring of the mercury is unavoidable, so that it is rendered highly sus-

ceptible to sickening. Californian millmen, though generally credited with advanced knowledge, often commit the serious blunder of adding heavy stones and lumps of iron to the charge, for the purpose of grinding the pulp finer to assist amalgamation, when in reality the grinding has as principal result an excessive flouring, sickening, and consequent waste of mercury.

The rotation of the barrel should be quite slow, varying, according to the size, from 12 to 16 rev. a minute, and continued for 8 to 24 hours, the larger size demanding the slower speed. When the agitation has sufficed, the pulp is further thinned by addition of water, and the speed is lessened, so as to facilitate concentration of the valuable contents into one mass, as much as possible ; and finally the charge is let out to pass over a succession of wells or riffles, and thence to a concentrating table of some form. On this latter is saved the more finely divided globules of mercury and the pyrites, for further treatment.

When the pyrites present is prone to decomposition and generation of sulphates, most harmful to amalgamation (see p. 208), an alkali is often added to the charge. This should be quicklime or a lye from wood ashes, but not the wood ashes themselves, if subsequent cyaniding of tailings is intended, as the charcoal accompanying wood ashes is most mischievous.

Occasionally an amalgamating barrel is used for amalgamating the gold freed from pyrites by roasting them, instead of having recourse to chlorination.

*Pans.*—The ordinary amalgamating pan, as made by Fraser & Chalmers, is shown in Fig. 79. A generally approved form is that with cast-iron bottom *a* and wooden sides *b*, diameter inside of staves being about 5 ft., but varying sometimes a few inches up to 5½ ft. The mullers *c* are of various designs, calculated to promote the most rapid circulation and intermixture of the pulp. In some districts, copper plates are introduced into the pan, and much of the amalgam is found attached to these ; but the most usual system is to employ settlers specially for the collection of the mercury and amalgam after the pans are discharged. Generally one settler is

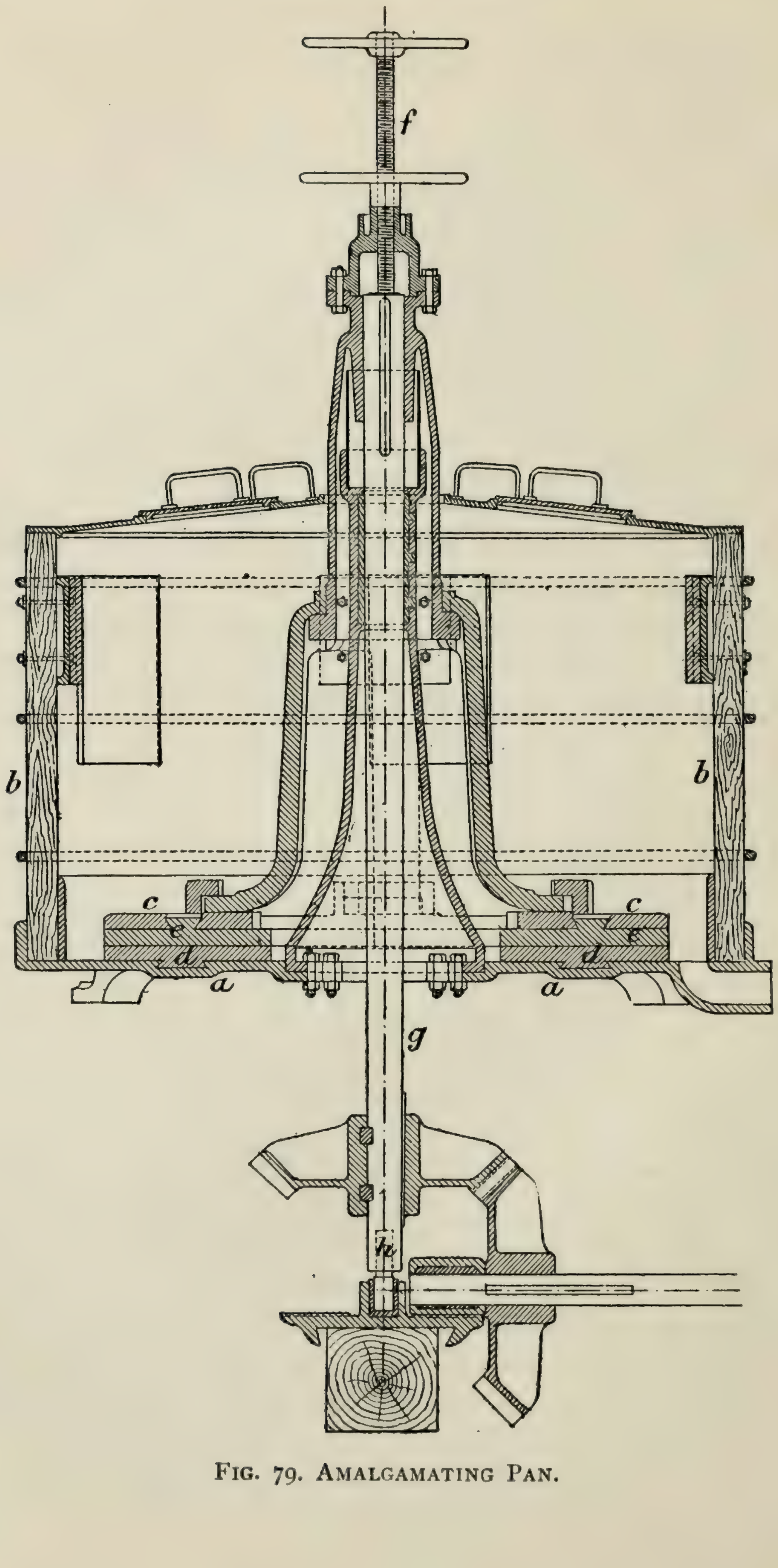



FIG. 79. AMALGAMATING PAN.

used to each two pans. While the pulp is being worked in the pans, steam is introduced either into a steam bottom, or direct into the pulp. The bottom *a* of the pan is usually protected by cast-iron dies *d*, and the muller is furnished with adjustable shoes *e* so that, if grinding the ore be necessary, the wearing surfaces are renewable. The muller is always adjustable by hand wheel and screws *f* on top of the spindle *g*, so that the shoes and dies can be brought together when grinding, or the muller can be raised above the dies for circulation and mixing only. The spindle is provided with a renewable steel toe, which is ground to perfect fit in spindle, and tempered. The step box is bushed with brass. Loose tempered-steel buttons are supplied with step box for spindle toe to rest upon. The speed of the pan muller is usually 65 to 75 rev. per minute.

This kind of pan, which is essentially of the Wheeler pattern, is used in Australia, California, and India, for grinding tailings and raw concentrates, and for amalgamating roasted concentrates.

The iron wearing surfaces (shoes and dies) should be as durable as they can be got, because the iron abraded from them is injurious to amalgamation, and causes loss of mercury. This has been very strikingly demonstrated by experiments with emery discs in place of iron, and is one of the main reasons why "improved" arrastras made of iron are much inferior as amalgamators to the old-fashioned stone-bedded article. G. M. Barber has described * a case where, owing to the great abrasion of iron from the grinding surfaces, through the extreme hardness of the ore, the gold particles not only were contaminated, but actually became incorporated with the ground cast iron and rendered unamalgamable ; and B. Kitto † has had similar experience, finding it quite impossible to separate the iron from the gold by a magnet, as the small particles of metallic iron picked up the fine gold just as effectually as if it were picked up with tweezers, but when the old pans were replaced by new ones made of proper hard material the difficulty was completely overcome.

* Trans. Inst. M. & M., v. 101 (1896).
† Ibid. v. 113 (1896).

The ordinary charge of mercury is 200 lb. per pan.

The custom of adding wood ashes to concentrates undergoing pan amalgamation needs some modification from ordinary practice if the tailings are to be subsequently cyanided, because, no matter how sifted, the ashes will always contain some charcoal, which causes a precipitation and loss of gold during cyanidation. Either charcoal must be efficiently removed, as for instance by leaching the alkali out of the wood ashes with hot water, and using only the filtered liquor, or the wood ashes must give place to another source of alkali, such as lime.

The consumption of water is about 120 gal. per hour for each pan, and 60 gal. per hour for each settler. Where sea water is the only kind available, and the gold at the same time is very fine, the loss by suspension may become excessive. This condition is encountered in the Guanaco mineral district of Chili, and it was found by G. M. Barber * that 10 to 20 % of the assay value of the ore was lost in this way. It was even suspected by him that some of the gold passed into actual solution, but this is not in accordance with what is known of the solvent action of salt water on gold, and probably the buoyancy of the water and lightness of the gold account for the whole loss.

A few working results will be interesting.

At a mill in Chili, the excessive loss of float gold is remedied by crushing dry and subsequently pan-amalgamating. The loss of mercury in the example quoted by Barber is about ·3 lb. per ton. The extraction,† calculated on over 15,000 t., assaying 36·8 dwt. fine gold per ton, averages only 75·84 %, so that about 9 dwt. per ton is lost in the tailings.

The principal Indian mines have long relied upon pan amalgamation for treatment of a portion of their tailings, but are now replacing it by cyanide. At the Champion Reef in 1898, the operations by Wheeler pan cost 9*s.* 1½*d.*, and the extraction was 3·94 dwt., per ton treated. The Mysore Co., in the same year, spent 7*s.* 9¾*d.* per ton, recovering 4·88 dwt.;

* Trans. Inst. M. & M., v. 102 (1896).

† Ibid., v. 106 (1896).

and the Nundydroog, 6*s.* 6*d.* for 2·37 dwt. Detailed costs are as follows :—

| | Mysore. | | Nundydroog. | |
|---|---|---|---|---|
| | *s.* | *d.* | *s.* | *d.* |
| Labour, European | 0 | 6·76 | | |
| ,, native | 1 | 4·72 | | |
| | 1 | 11·48 | 1 | 11 |
| Fuel | 3 | 3·31 | 3 | 0 |
| Shoes and dies | 1 | 10·71 | | |
| Mercury | 0 | 3·00 | | |
| Oils and tallow | 0 | 1·85 | | |
| General stores | 0 | 3·39 | | |
| | 2 | 6·95 | 1 | 7 |
| Total | 7 | 9·74 | 6 | 6 |

The latest champion of pan amalgamation is T. G. Davey, who describes * Australian, or more particularly, Victorian practice.

Water is first introduced into the pan from a convenient tap, to a depth of about 4 in. The muller is then set in motion, and the ore is gradually charged to within 2 in. from the top of the wings or splash-plates, and water or ore is then added until a consistence resembling that of mortar is attained. About 20 lb. lime, 1½ oz. potassium bichromate, and 2 oz. caustic soda should be mixed with the (roasted) ore before it is put into the pan. The muller is then lowered, and grinding proceeds for 4 to 6 hours, or until the pulp has been reduced to a state of almost impalpable slime. At this stage the muller should be raised about 1 in. from the bottom, and a quantity of mercury, ranging from 20 to 50 lb., according to the gold contents of the ore, is added to the charge in the form of spray by squeezing it through a wet calico bag.

Should all the conditions be favourable to amalgamation, the whole of the gold will be taken up or absorbed by the mercury in about ½ hour. To ascertain whether this is the case, a small dishful of the pulp should be taken out and carefully "panned off." Should amalgamation be complete, no free gold will be seen, and the mercury will be

* Trans. Inst. M. & M., viii. (June 20, 1900).

mostly in one bright, rounded, and "lively" mass, with perhaps a few minute but bright globules, which may be easily collected by friction. On the contrary, should the conditions be unfavourable, the gold, in spite of the presence of mercury, will form an independent "head" in the "panning-off" dish, and the mercury will be found either in a bright globule, partially coated with heavy particles of ore, or perhaps metallic iron, from the wearing of the grinders, or in elongated, dull and dirty strips, which in spite of rubbing will show no affinity for gold.

"The use of potash bichromate is to subdue the evolution of hydrogen gas on the surface of the mercury, caused by the action of any free sulphuric acid which may be present in the pan due to the decomposition of sulphates. The presence of too much circulating hydrogen produces such a violent action (rotary motion) on the surface of the globules of mercury that even gold is rejected by them, and thrown off at a tangent, although the mercury would be perfectly clean and bright. On the other hand, in the absence of free sulphuric acid, the addition of potash bichromate 'sickens' the mercury, rendering it dull and sluggish, and therefore unfavourable to amalgamation with gold.

"It is obvious, therefore, that the happy medium should be aimed at, that is, to bring about a quasi-quiescent evolution of hydrogen on the surface of the mercury, sufficient to render each particle absolutely bright and globular, without producing such violent action as to cause it to repel any gold which might come into contact with it. In a word, the presence of a minute excess of acid is necessary to perfect amalgamation.

"The recovery of floured mercury is rendered practicable by the adoption of this mode of treatment."

The advantages to be derived from the use of the reagents referred to are deemed to have been demonstrated by experiments carried out by Davey on a working scale some years ago.

The nett results of these experimental runs are said to be :—

(*a*) 82 % recovery of gold and 96 % of mercury: contents not stated.

(*b*) 85 % of gold and over 100 % of mercury, the excess being derived from old blanketings: assay, 5 oz.

Inasmuch as thorough roasting is insisted on as a preliminary, and grinding to slimes is effected in the pans, it is difficult to see how the cost can be less than in chlorination, while the extraction is not nearly as complete.

An up-to-date American "combination" mill, at the Eureka Hill, Utah, managed by Arthur Buckbee, is described by W. McDermott.* The pulp from the stamps passes through hydrometric sizers to Frue vanners, and thence to 5-ft. amalgamating pans and 8-ft. settlers, there being 32 of the former and 16 of the latter for 100 stamps. The vanner tailings go to settling tanks above and behind the pans, the overflow water being pumped back to the stamps, as it is not entirely clear. The settled pulp is dumped by trap-doors in the bottom of the tanks on to a lower floor, which is level with the tops of the pans. Into these it is charged in lots of about 3000 lb., and sufficient slimes are added to produce the proper consistence. With each charge about 5 % salt, 3 lb. copper sulphate, 2 lb. sulphuric acid, 2 lb. iron borings, and 2 oz. concentrated alkaline lye, are introduced. Steam is turned in, raising the temperature to about 180° F., which occupies some 2 hours. Then 200 lb. mercury is added. The pan is run for a further 6 hours, making 8 hours in all, and the contents are discharged into a settler, where water is added, and the amalgam is settled and drawn off through siphon taps. The average extraction is about 80 % on ore containing originally 20 oz. silver, 2 dwt. gold, 4 % lead, and a little copper, about half the value being taken out by the vanners.

The Hungarian amalgamating bowl, called also the Tyrolese mill, is shown in Fig. 80. The basin *a* is of ½-in. cast iron, 6½ in. high, 24 in. diam. at top and 18 in. at the flat bottom inside. An iron pipe *b*, rising for 4 or 5 in., is cast to the centre of the basin; it has 3-in. outer and 1½-in. inner diam.,

* Trans. Inst. M. & M., vi. 245 (1898).

and receives the lower pivot bearings *c* of the spindle *d*. These bearings *c* are usually of cast iron, and are protected against admission of sand, etc., by a mantle or hood of sheet iron *e*. About 3½ in. above the bottom is a hole, 4 in. wide, in the rim of the basin, to which a tin or sheet-iron lip *f* is riveted. The bowl is fixed to the floor, either by screws through two lugs cast opposite to each other on its base, or the latter is made with a projecting rim, over which iron cleats are driven into the planking. The runner *g* is made of pine-wood, of the exact shape of the inside of the basin, but slightly

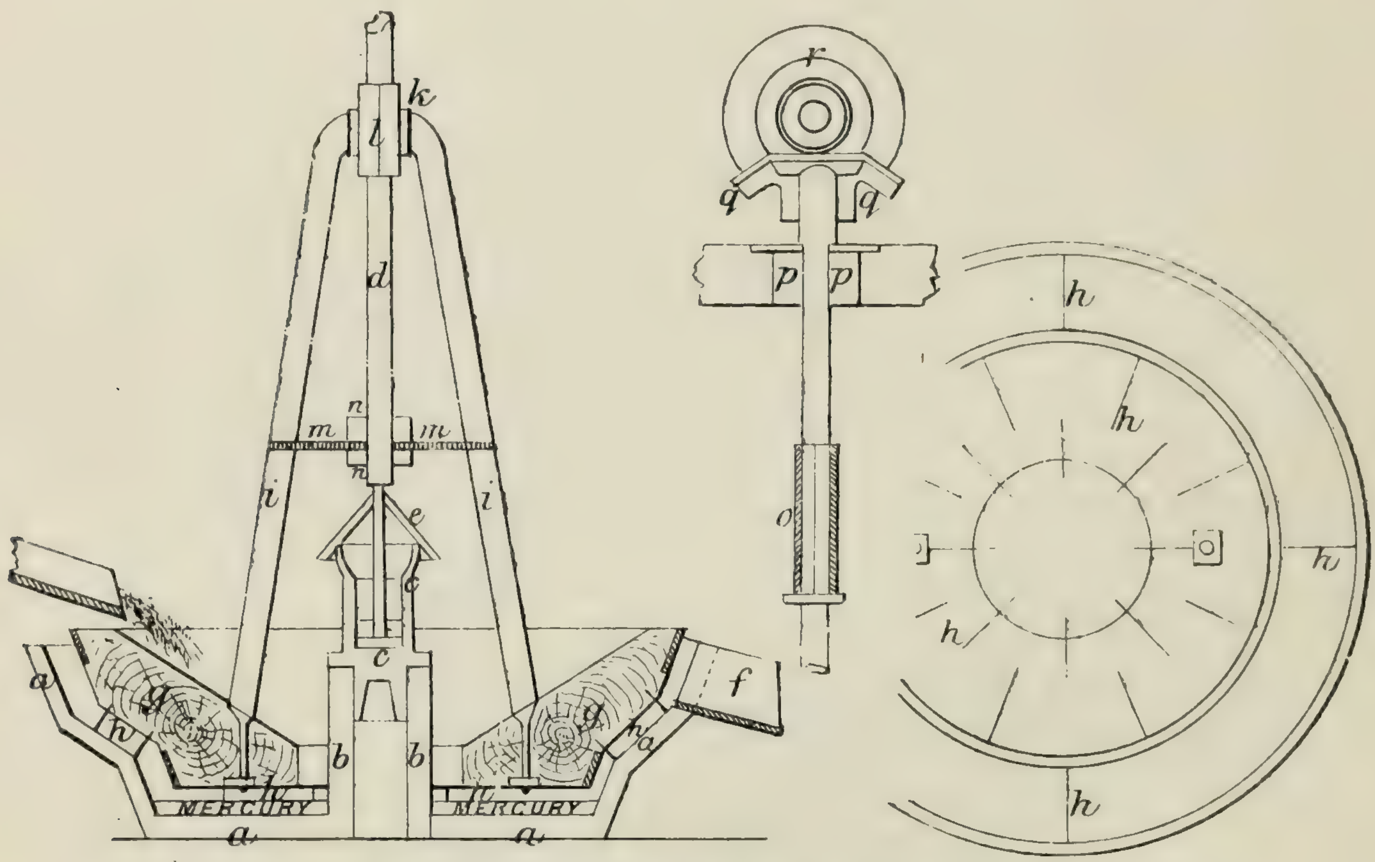



FIG. 80.—HUNGARIAN BOWL OR TYROLESE MILL.

smaller, and has at top a wide funnel-shaped cavity, which communicates with a cylindrical hole, 5 in. diam., through the centre. Its size and shape are such that, when suspended centrally in its proper position in the basin—the hole just mentioned allowing it to go freely over the cylinder above described—its surface is parallel to the inner surface of the basin, and leaves an open space ¾ in. wide round the side and 1½ in. at the bottom, and it projects about 1 in. above the top of the basin. Thin sheet-iron hoops (2) round the circumference protect it against cracking, and its bottom is armed

with about 20 sheet-iron blades or scrapers *h*, 2½ in. long and ½ in. thick, which are radially driven into the wood so as to project exactly ¾ in. The runner is attached to the spindle by the wrought-iron fork *i*, its prongs going through the runner, and being screwed tight, while a square collar *k*, formed at their junction, is slid over a square portion *l* of the spindle; 6 or 7 in. below this the prongs are joined by an iron crossbar *m*, which fits with a flat round collar over the spindle, where the latter has a screw-thread cut for several inches. Nuts *n*, above and below the collar, serve for both fixing and adjusting the runner. A little above the upper collar each spindle has a coupling for throwing the runner out of action independently of the other mills. Motive power is preferably applied through mitre gearing, as shown at *p q r*, the bearings being of white beechwood. The runners should make 16 to 20 rev. per minute, and 25 mills require only 1 h.p.

The charge of mercury for each mill varies from 30 to 50 lb., and forms a ring 3 to 7 in. wide and ½ in. thick. With low grade ores, such as prevail in Hungary, the mills are cleaned up only once a month. The capacity of each mill is about 54 cub. ft. of pulp and water per hour, the solids in which will not exceed 1 to 1½ cwt. The loss of mercury ranges from 3½ to 8 dwt. per ton, according to the amount of galena in the ore. The mills are arranged in two tiers, the upper delivering to the lower, and two mills are required for every ton of ore crushed.

A machine closely allied to the Hungarian bowl has been for many years in use at the Pestarena Co.'s mills, Italy, and the following figures are derived from the official report for 1898. An average of 25·8 mills, working for 307 days, passed through 5903 t. of dry ore, or less than ¾ t. per mill per diem. The fineness of the bullion is 747·9 gold and 242 silver. Assay contents are 14·47 dwt. fine gold per ton, and extraction is 12·27 dwt., or 84·8 %. Consumption of mercury is 157·136 dwt. per oz. of bullion won, or 128·92 dwt. per ton milled, and the cost for that alone is 10·63*d.* per ton.

Much ingenuity has been displayed in the invention of various contrivances for forcing mercury vapour, with or with-

out steam, into contact with the gold contained in ore, either at the moment of its exit from the battery, or as it emerges from the roasting furnace ; but all have resulted in no practical advantage, and in great risk of injury to the workmen by escaping fumes.

As a clean-up pan, the little Berdan is much to be preferred to any other. In consists of an open basin, about 2 ft. diam. and 10 in. deep, set at an angle, and made to rotate slowly. A drag is so suspended that it constantly rubs against the upper side, and a jet of clean water is led into the pan. A sufficiency of fresh mercury is placed in the pan with the dirty amalgam from the clean-up, and under the influence of rubbing and clean water the impurities are gradually eliminated and washed over the lower margin of the pan, passing into the tailings launder. The clean mercury absorbs the freed amalgam, and being always out of reach of the drag, is not broken up as it would be by a ball running in the lower half of the pan.

**Retorting Amalgam.**—For small undertakings, a very efficient amalgam retort can be made out of an empty mercury bottle. About 3 in. of the upper end is cut off, and the two faces are made as true as possible, by planing where that is feasible, and the joint being recessed inwards slightly. A strong clamp and wedges suffice to retain the lid. Into the threaded aperture which receives the plug when the flask is used for holding mercury, is inserted a bent piece of gas piping, by which the volatilised mercury passes to the condenser. Bucket-shaped retorts of various capacities, on similar lines, can also be purchased from the makers. These small retorts can be hung over an open fire made on the ground, or in an assay furnace.

Large mills adopt a cylindrical retort, built into a proper fireplace, such as that shown in Fig. 81. The retort *a* is enclosed in a fire-brick furnace *b*, so constructed that the flames shall have full play upon it. The cylindrical form of the retort enables it to be turned round when one side becomes partially burned. A common size is 5 ft. long and 1½-in. metal, the

inside cylindrical portion being 3 ft. long and 1 ft. diam. It is cast with a gradual contraction towards the neck *e*, which is $2\frac{1}{2}$ in. diam., and is furnished with a flange *f*, to which the condensing pipe *g* is bolted. This last bends downwards into a water-jacket *h*, the inflow of cold water at *i* and exit of heated water at *j* being continuous, and thus ensuring the coolness necessary to condensation of the volatilised mercury. The amalgam is placed in the retort in sheet-iron trays fitting the cylinder. The flue *c* leads to chimney *d*; *k* is the hot-water effluent.

While apparently a very simple process, the retorting of amalgam demands the observance of certain precautions. In the first place, it is necessary to prevent adherence of the bullion to the sides of the retort, and to this end the interior of the latter is coated with a layer of chalk or whiting in ordinary practice, though Prof. Louis recommends a thin paste of equal parts of finely ground fireclay and graphite. Sometimes a few sheets of notepaper are substituted, the very thin layer of ash sufficing to prevent intimate contact. The balls of amalgam, about the size of small oranges, should be packed in loosely, leaving air spaces between them, and not be rammed in tightly, as is sometimes done. In very large retorts the contents are kept from uniting into a single cumbrous mass, by interposing sheet iron perforated trays and partitions, though sometimes sheets of paper are relied on.

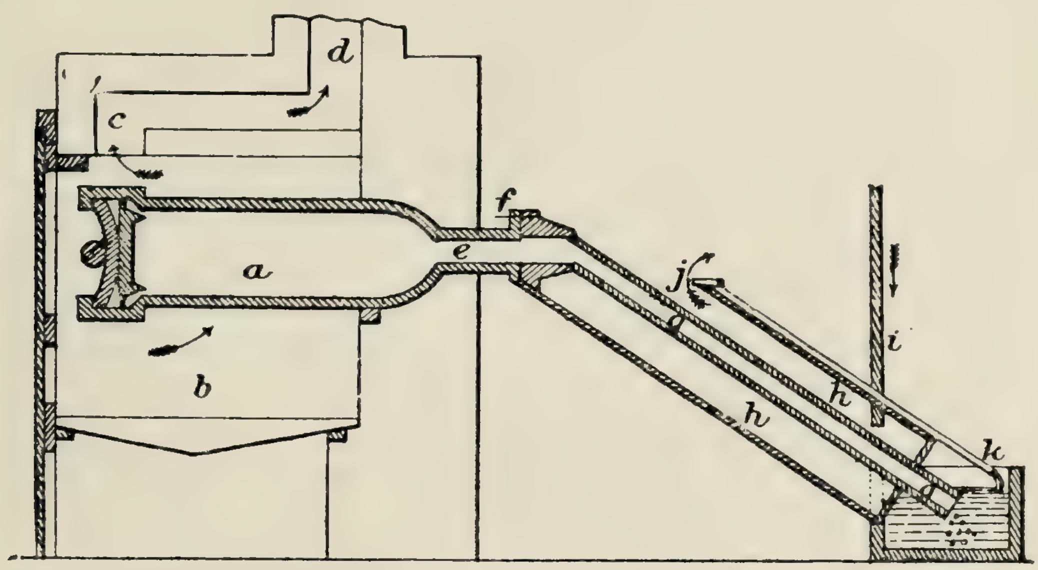



FIG. 81.—AMALGAM RETORT.

The luting on of the lid is a most important matter, or great loss of mercury fumes may occur, with disastrous consequences to the operator. Common clay is sometimes used as a lute, with or without the addition of wood ashes, and worked into a stiff paste. Fireclay is better, and in this connection it may be mentioned that old assay crucibles, ground down in a Berdan pan, answer excellently. Prof. Louis would add graphite, as for the lining wash. The lid is very forcibly held in place by a clamp, a bale, a set-screw, or bolts.

Fire must be applied gradually until volatilisation commences, be maintained with regularity so long as it lasts, and finally, when it has apparently ceased, the heat must be increased till dull redness is reached, and continued for a few minutes. The fire may then be allowed to die down, and the retort to cool steadily. In large mills, it is a usual practice to start the retort after midday, finish retorting by the evening, and leave the retort and contents intact till next morning, by which time they will have cooled sufficiently to make handling the bullion a less irksome matter, and by the same delay the retort is not subjected to undesirably rapid change of temperature. The heating of the retort must be as evenly distributed as possible, and it is most important that it shall commence from above and gradually extend downwards. This obviates all risk of volatilisation commencing from below and being obstructed, which may easily lead to an injury being done to the retort. Except for the last few minutes, the heat should not be allowed to exceed the demands for steady volatilisation.

An unduly great heat will soften the iron and cause it to sag under the contained weight; and, especially when the bullion is impure and the cast iron not too good, may bring about incipient fusion of the surfaces of the bullion and the retort, so that the latter becomes sensibly impregnated with gold. The author has seen a retort which had been overheated and burned, and which, on being pulled out and broken up, was found literally saturated with gold for over ½ in. of its thickness in the burned area; a good many ounces of gold were recovered from it by dissolving away the iron in acid

As a rule, the downward heating of the retort is not sufficiently attended to, and in the example shown in Fig. 81, no provision is made for firing from above, as there should be ; with a fire-place beneath only, as indicated, the under-side of the retort must necessarily always be the hottest, which is not right.

Freedom of escape for the volatilised mercury must be carefully secured. For this reason, it is not advisable to fill the cylindrical retort above the level at which issue of vapour takes place. With foul bullion particularly this is a wise precaution, as other volatile metals which solidify on cooling, and will therefore not flow away as the mercury does, soon accumulate in the neck *e*, diminishing its area, and impeding the escape of the mercury, so that leaks and even fractures of joints may be caused by the pressure generated.

Condensation of the volatilised mercury must be entirely completed inside the condensing pipe, and the liquid metal must escape freely and drop through air into a vessel containing water placed to receive it. In Fig. 81, the condensing pipe is shown immersed in water at the lower end ; this is radically wrong, and must be most carefully avoided, for the reason that a cooling of the retort might produce a vacuum, which would result in cold water being drawn into it, with probable disastrous consequences. A very simple and effective arrangement is to lay the condensing pipe in a trough of sheet iron or common galvanised guttering, covering it with strips of old blanket kept constantly soaked with cold water from a continuous supply, such as a small pipe leading from an overhead cistern. A bit of blanket may also be loosely hung over the mouth of the condensing pipe. Below the discharge, an enamelled bucket containing water receives the issuing mercury. Provision must be made for conveying away the cooling-water.

Retorting operations on a small scale are often conducted out of doors in an improvised arrangement, but at mills of any importance, proper accommodation is provided. The retort room may with entire convenience be combined with the melting room and assay office, the retort, bullion furnace, reducing furnace,

and muffle being arranged side by side against the same wall, and connecting with the same settling flue and chimney stack. The brickwork must be well bound and tied with plates and rods to resist strains from heat, and all fireplaces and furnaces should be of best firebrick, using ordinary brick only where no fire is encountered. The floor of this room should be most carefully laid in concrete faced with best Portland cement, and slightly dished to a small runnel leading into a deep trap, in which an enamelled-iron bucket should constantly stand, so as to give a minimum of trouble in catching and recovering accidentally spilled mercury and other valuable matters. The floor should be carefully washed down after operations, or preferably every day, and the residues collected in the bucket may be periodically ground and washed in a Berdan pan, and finally amalgamated with a little fresh mercury in a similar machine.

It is the usual custom in retorting to make no addition to the amalgam, but at the Crown Deep, according to F. L. Carter,* a little nitre is introduced with it, for the purpose of oxidising some of the base metal, such as copper and lead, and making the bullion finer at this stage.

The yield of crude bullion or "sponge" ranges between 25 and 45 % of the weight of amalgam retorted; a very general figure is about 33 %. It depends chiefly on the dryness of the amalgam, that is to say, the completeness with which excess mercury has been squeezed out of it, and upon the fineness of the gold particles, coarse gold giving the highest percentage.

The results obtained at Lucknow (N.S.W.) on 4 years product (1896–9) of amalgam, amounting to 51,442 oz., were 23,726·75 oz. sponge gold, or 46·12 %; this, on melting, was reduced to 21,389·71 oz., or 41·58 %; and again, on refining to 20,500·061 oz., or 39·85 %. The fineness of the ultimate bullion was 856 gold, and 144 silver.

The cakes of sponge must be cut up with hammer and chisel, if necessary, to reduce them to a suitable size before transference to the melting-pot.

* En. & Min. Jl., Nov. 12, 1898.

**Practical Results.**—A few figures, showing the results attained by amalgamation methods in practical working cannot fail to be interesting.

*Extraction.*—In a presidential address, it was stated by J. H. Collins * that while occasionally a 90 % extraction is reached when the gold is free and very coarse, less than 60 % results when the gold is extremely fine, or the veinstone contains large quantities of clay, iron oxide, barytes, or manganese oxide, and he has known as little as 20 % to be saved by amalgamation. On the whole, he doubts whether an average of 70 % has been reached by this method.

The Thames field, New Zealand, scarcely exceeds 50 %, and the big Indian mills are only credited with about 72 %.

AMALGAMATION : EXTRACTION BY.

| —— | Assay. | Tailings. | Extraction. | Extraction. | Remarks. |
|---|---|---|---|---|---|
| Brazil : | dwt. | dwt. | dwt. | % | |
| Faria . . . . | .. | .. | 4·98 | .. | Battery. |
| Morro Velho . . | .. | .. | 16·14 | 73·12 | ,, |
| Ouro Preto . . | 5·70 | ·503 | 5·197 | 81·30 | ,, 1898 |
| Do. . . | 5·77 | ·419 | 5·351 | 82·67 | ,, 1899 |
| St. John del Rey . | .. | .. | 16·47 | 79·75 | ,, |
| California : | | | | | |
| North Star . . | .. | 1·000 | 6·14 | 86·00 | 2·57 dwt. in concentrates. |
| W.Y.O.D. . . | .. | 1·100 | .. | 87·50 | |
| Chili . . . . . | 36·8 | 8·900 | 27·90 | 75·84 | Pan. Fine gold. |
| Colorado : | | | | | |
| Hidden Treasure . | .. | .. | 7·42 | 70·40 | Gold. Battery. |
| | .. | .. | .. | 39·70 | Silver. ,, |
| India : | | | | | |
| Champion . . | .. | .. | 27·54 | .. | Battery and pan. Fine gold. |
| Mysore . . . | .. | .. | 29·57 | .. | Battery. |
| | .. | .. | ·72 | .. | Pan ; tailings. |
| Nundydroog . . | .. | .. | 17·11 | .. | Battery. |
| | .. | .. | ·54 | .. | Pan ; tailings. |

* Trans. Inst. M. & M., iii. 317 (1895).

AMALGAMATION: EXTRACTION BY—*continued.*

| —— | Assay. | Tailings. | Extraction. | Extraction. | Remarks. |
|---|---|---|---|---|---|
| India (*contd.*): | dwt. | dwt. | dwt. | % | |
| Ooregum . . . | .. | .. | 11·78 | .. | Battery. |
| | .. | .. | 2·51 | .. | Pan; tailings. |
| Italy: | | | | | |
| Pestarena . . . | 14·47 | 2·20 | 12·27 | 84·80 | Pan. Fine gold. |
| New South Wales: | | | | | |
| Lucknow . . . | 31·2 | 4·68 | 26·52 | 85·00 | 16·64 dwt. in concentrates, 1897. |
| Do. . . . | 30·8 | 5·04 | 25·76 | 83·63 | 12·72 dwt. in concentrates, 1898. |
| Do. . . . | 13·52 | 2·44 | 11·08 | 81·95 | 6·04 dwt. in concentrates, 1899. |
| Transvaal: | | | | | |
| Angelo . . . | 17·48 | 9·02 | 8·46 | 48·00 | Battery. |
| Bonanza . . . | 26·77 | 10·76 | 16·01 | 60·00 | ,, |
| City . . . . | 8·00 | 3·63 | 4·37 | 54·70 | ,, |
| Do. . . . . | 7·00 | 3·06 | 3·94 | 56·40 | ,, |
| Crown . . . | 8·00 | 3·39 | 4·61 | 57·74 | ,, |
| Crown deep . . | 11·21 | 5·85 | 5·36 | 48·00 | ,, |
| Driefontein . . | 12·16 | 6·88 | 5·28 | 43·50 | ,, |
| Durban Roodept. | 12·85 | 5·40 | 7·45 | 58·00 | ,, |
| Ferreira . . . | 21·00 | 8·40 | 12·60 | 60·00 | ,, |
| Geldenhuis . . | 7·00 | 2·52 | 4·48 | 64·00 | ,, |
| Glen Deep . . | 11·96 | 5·61 | 6·35 | 53·50 | ,, |
| Jubilee. . . . | 10·50 | 3·15 | 7·35 | 70·00 | ,, |
| Jumpers Deep . | 13·31 | 6·53 | 6·78 | 51·00 | ,, |
| Langlaagte Deep. | 9·93 | 4·63 | 5·30 | 53·50 | ,, |
| May . . . . | 7·25 | 2·47 | 4·78 | 66·00 | ,, |
| New Comet . . | 10·84 | 5·62 | 5·22 | 48·25 | ,, |
| New Heriot . . | 9·25 | 3·52 | 5·73 | 62·00 | ,, |
| New Primrose . | 6·75 | 2·64 | 4·11 | 61·00 | ,, |
| Nourse Deep . . | 11·34 | 5·60 | 5·74 | 50·50 | ,, |
| Paarl . . . . | 8·36 | 4·06 | 4·30 | 51·50 | ,, |
| Robinson . . . | 18·50 | 11·45 | 7·05 | 60·00 | ,, |
| Rose Deep . . | 11·99 | 5·76 | 6·23 | 52·00 | ,, |
| Witwatersrand . | 9·80 | 4·43 | 5·37 | 55·00 | ,, |
| Victoria: . . . | 8 to 9 | 1 to 1½ | 7 to 7½ | 85·00 | ,, |
| Clunes . . . . | 4·70 | ·75 | 3·95 | 84·00 | ,, |

The consumption and cost of mercury (reckoned at 1·66*d.* per oz.) used in amalgamation at a number of mills are shown in the annexed table :—

MERCURY CONSUMPTION AND COST.

| — | Mercury Lost. | | Cost per ton Milled. | Remarks. |
|---|---|---|---|---|
| | Per ton Milled. | Per oz. Bullion. | | |
| California : | dwt. | dwt. | *d.* | |
| Empire . . . . . . | 9·00 | .. | ·747 | Battery. |
| Gover . . . . . . | 2·60 | .. | ·215 | ,, |
| Idaho . . . . . . | 13·00 | .. | 1·079 | ,, |
| Keystone . . . . . | 6·50 | .. | ·529 | ,, |
| North Star . . . . | 14·50 | .. | 1·680 | ,, |
| Wildman . . . . . | 4·50 | .. | ·373 | ,, |
| W.Y.O.D. . . . . | 11·00 | .. | ·913 | ,, |
| Chili . . . . . . . . | 87·00 | 18·67 | 7·221 | Pans. |
| Colorado : | | | | |
| Gregory Bobtail . . . | 5·20 | .. | ·431 | Battery. |
| Hidden Treasure . . | 4·30 | 11·59 | ·357 | ,, |
| New York . . . . | 3·70 | .. | ·307 | ,, |
| Prize . . . . . . . | 9·70 | .. | ·805 | ,, |
| Randolph . . . . | 9·80 | .. | ·813 | ,, |
| Dakota : | | | | |
| Homestake . . . . | 5·00 | 22·22 | ·415 | ,, |
| Italy : | | | | |
| Pestarena . . . . . | 128 92 | 157·136 | 10·633 | Pans. |
| Mexico : | | | | |
| Mesquital . . . . . | .. | 12·75 | .. | Battery. |
| New South Wales : | | | | |
| Lucknow . . . . . | 4·14 | 10·58 | ·343 | ,, |
| New Zealand : | | | | |
| Phœnix . . . . . . | 7·50 | .. | ·622 | ,, |
| Premier . . . . . . | 6·25 | .. | ·518 | ,, |
| Reliance . . . . . | 4·46 | .. | ·370 | ,, |
| Saxon . . . . . . . | 12·95 | .. | 1·074 | Battery and pans. |
| Norway . . . . . . | 5·95 | .. | ·494 | Battery. |

MERCURY CONSUMPTION AND COST—*continued.*

| — | Mercury Lost. Per ton Milled. | Mercury Lost. Per oz. Bullion. | Cost per ton Milled. | Remarks. |
|---|---|---|---|---|
| Transvaal : | dwt. | dwt. | *d.* | |
| Geldenhuis . . . . . | .. | .. | ·334 | Battery. |
| Witwatersrand . . . | .. | .. | ·199 | ,, |
| Victoria : | | | | |
| Britannia . . . . . | 2·32 | .. | ·192 | ,, |
| Catherine . . . . . | 6·52 | .. | ·541 | ,, |
| Clunes. . . . . . . | 0·24 | 1·22 | ·019 | ,, |
| ,, . . . . . . . | 2·68 | .. | ·222 | Battery and barrels. |
| Fortuna . . . . . | 7·85 | .. | ·651 | Battery. |
| Harrietville . . . . | 7·14 | .. | ·592 | ,, |
| ,, . . . . | 16·96 | .. | 1·407 | Battery and pans. |
| Hillsborough . . . . | 3·57 | .. | ·296 | Battery. |
| New Chum . . . . | 6 92 | .. | ·574 | ,, |
| New Normanby . . . | 5·00 | .. | ·415 | ,, |
| North Cornish . . . | 4·37 | .. | ·362 | ,, |
| Oriental . . . . . . | 7·14 | .. | ·592 | ,, |
| Pearl . . . . . . . | 6·00 | .. | ·498 | ,, |
| Railway . . . . . . | 7·14 | .. | ·592 | ,, |
| South St. Mungo. . . | 8·48 | .. | ·703 | ,, |
| Stephens . . . . . . | 7·14 | .. | ·592 | ,, |
| West Australia : | | | | |
| North Boulder . . . | .. | .. | 1·550 | ,, |

# CHAPTER VI.

## DRY MILLING.

WET milling of gold-bearing stone is now so general, that the adoption of dry milling is regarded in the light of a novelty. But as a matter of fact, dry milling is by far the older method, and it only by degrees gave way to wet milling, when it was found that water assisted in clearing the mortar of pulp, and served as an aid to the separation of the valuable from the valueless portions after crushing was completed.

Much difference of opinion has been expressed regarding the respective merits of wet and dry milling, advocates of the one method condemning the other in a most comprehensive manner. Manifestly this is absurd. No claim of absolute and universal superiority can be substantiated by either, the simple truth being that each has its advantages. In certain cases, preference is justly given to wet milling; in a second class, dry milling is unquestionably more suitable; while in a third class, there is much to be said in favour of both, and selection is a matter of difficulty. Occasionally a mine produces two kinds of ore, the one demanding wet treatment while the other requires dry, yet the output does not justify the erection of two systems of milling. Such are among the problems which only wide knowledge and experience, and a most careful appreciation of local conditions, can satisfactorily grapple with. Quite an education is to be gained from studying the many instances, on all the principal gold-fields, of wet-crushing mills being converted into dry, and *vice versâ*. This condition of things is going on even now, showing how much still remains to be learned on the subject, and what a tendency exists to copy some other plant, instead of working out each

proposition on its own merits before coming to a decision. Many a good mine has been irrevocably damned by an unsuitable mill and ignorance of its milling requirements.

While the stamp-battery continues to hold a pre-eminent position among wet-crushing machinery, it has been superseded by specially-designed appliances for dry-crushing, notable among which are rolls and ball mills, though dry batteries survive in places. It is claimed for rolls and ball mills that they cost less, are more easily and quickly erected, require less power, and produce a pulp of more consistent size, than wet stamp-batteries ; on the other hand, they are much more liable to injury, suffer greater wear and tear, generally require the ore to be dried, and, except where unusual precautions are taken, permit a loss of fine dust which is exceedingly injurious to the workmen. However, none of these considerations is really material to the question of choice between wet and dry crushing : that must be determined by local conditions (especially as to water and fuel), by the nature and peculiarities of the ore to be treated, and by the character of the treatment which is to follow. The reduction of the ore in size, whether by wet or dry process, must be regarded as a mere preliminary to the extraction of its value. The creation of dust is a serious feature, both in actual loss of ore and in causing sickness among the workmen, besides injury to machinery ; only in the best installations is this remedied, and, where not prevented, it is a greater evil than the sliming which forms such a heavy indictment against wet stamping.

It may pretty safely be said, despite the economies claimed for dry crushing, that it is not had recourse to except when circumstances compel its adoption, and then practically only where direct cyaniding or chlorinating is to follow. Amalgamation has been attempted on dry-crushed gold ores in bulk on but rare occasions ; and though several forms of concentrator operating by means of air have been brought to a high degree of efficiency, their use is very exceptional.

Scarcity of good water is a great factor in giving a status to dry crushing. According to Franklin White,* at the Luipaard's Vlei mine, Transvaal, they were able, with dry milling,

* Trans. Inst. M. & M., vii. 61 (1898).

to accomplish cyaniding with 24 gal. of water to the ton of ore; whereas, in a wet mill, at least 7 times as much water had to be used, while there was a consumption of power in raising that water to the battery tanks, and again in elevating it with the tailings.

In Western Australia, dry crushing is much in vogue, because the water supply is limited and very impure, the presence of solid matters in suspension and salts in solution opposing great obstacles to the treatment which follows the crushing.

The question of sliming is in some cases regarded as of paramount importance, because of the cost and difficulty entailed in leaching slimes, and the loss of fine gold sustained if the slimes are neglected. Differences in the physical nature of ores have much to do with the amount of sliming which they suffer under various systems of pulverisation, and it is impossible to say offhand regarding any ore that it will crush best (that is to say, give a maximum product of the desired gauge and a minimum product of less than the desired gauge), wet or dry, under heavy stamps or light, in rolls or in ball mills. Only actual experimental treatment can determine the point. Very few such experiments have been recorded, and fewer still from which bias is absent.

One of the most recent and reliable tests was made by Franklin White, on Transvaal banket * resulting in the following figures :—

| Machine. | Screen Mesh. | Screen Mesh. | Percentage of pulp passing one screen and remaining on the next finer. | | | |
|---|---|---|---|---|---|---|
| | | | 30 × 30 | 40 × 40 | 60 × 60 | 90 × 90 |
| | per sq. in. | per lin. in. | | | | |
| Wet stamps . | 900 | 30 | 5·60 | 12·66 | 17·58 | 64·16 |
| Do. . | 700 | 26·5 | 11·15 | 28·53 | 9·21 | 51·11 |
| Dry stamps. . | 400 | 20 | 20·30 | 9·80 | 21·80 | 49·10 |
| Rolls . . . | 400 | 20 | 26·63 | 33·99 | 13·06 | 26·30 |
| Do. . . . | 500 | 22·4 | 9·30 | 41·85 | 15·38 | 33·47 |
| Ball mill . . | 500 | 22·4 | 20 17 | 24·38 | 13·88 | 41·67 |
| Niagara mill . | .. | .. | 20·17 | 24·30 | 24·30 | 31·23 |

* Jl. Chem. & Met. Soc. S. Africa, Jan. 15, 1898.

At first sight, this table makes out a very strong case against wet stamps, showing 50 to 60 % of slimes, whereas rolls produce only about half as much. But while useless as a guide to the treatment of any other ore than that operated on, the usefulness of these figures, even with reference to that, is very much impaired by the employment of a variety of screens instead of one standard mesh throughout. Comparison between the work of stamps at 700 and 900 mesh and that of rolls at 400 and 500 is no comparison at all, but totally misleading. One has only to compare the results of rolls at 400 and rolls at 500 mesh to see this. The increase of slimes cannot be computed on a reliable basis with varying mesh, because it is not in direct ratio to the increased number of holes per in. Thus rolls at 400 make 26·3 % of slimes, and at 500 they make 33·47 %; the direct proportion would be 32·8 %, and if the holes per linear inch (20 and 22·4 respectively) be adopted in the calculation, the ratio of 20 to 22·4 is as 26·3 % to only 29·45 % instead of 33·47 %. Now the direct relation of 26·3 % at 400 is 46·02 % at 700, and 59·17 % at 900, being about 2½ % less than was actually observed. That 2½ % is not greater than the progressive increase due to finer crushing, even if it is actually as great. The evidence therefore, is quite inconclusive; most certainly the case against wet stamps is not proved, and, inasmuch as no attempt is made to show that the wet-stamp mortar used was designed to give maximum facility of issue, it may even be that the trials in reality prove the superiority of wet milling. At any rate, so far as they go, they add nothing definite to our knowledge on the point.

The nature of the ore sometimes (as it is thought) compels dry crushing. This has been notably the case at some of the poorer mines of South Africa—Kantoor, Lydenburg, Swaziland, Klerksdorp, and even on the Rand itself; as also at Waihi (New Zealand), Mount Morgan (Queensland), the Mercur mine (Utah), and in the Kalgurli field (W. Australia). Thus it was found by actual test at the Lisbon Berlyn * mine, that wet crushing and amalgamation recovered only 1 to 1½ dwt.

* M. T. Brown, Trans. Inst. M. & M., vii. 50 (1898).

out of 8 to 9-dwt. ore, and produced 40% of slimes unfit for subsequent leaching. At Mount Morgan (Q.), calcining is rendered compulsory by presence of tellurium, and to effect dehydration of the limonite and kaolin in the ore. And so on, as will be described presently; but it would also seem that, in some of these cases, it has been too hastily assumed that dry milling is compulsory.

**Drying.**—Very few mines afford all their ore in a fit state for dry milling, and the application of some form of drying is practically an essential preliminary. No doubt the dehydration thus brought about is of advantage in subsequent leaching operations, as the particles of ore are rendered much more porous and absorptive; but it should be done in a careful and well regulated manner, avoiding any excess of temperature beyond that necessary for the purpose, or it may easily be productive of more harm than good, by causing incipient fusion—if not of the gold (which is hardly likely), at any rate of the pyritous bodies enveloping it, and thus little accretions of very rich material may be formed, which are certain to escape solution in ordinary leaching liquors.

It is obvious, therefore, that the use of a crude arrangement for drying ore is to be condemned on the score of irregularity, besides being equally objectionable on the score of cost both in labour and in fuel. Nevertheless, most ill-contrived plant has been widely used in New Zealand, and is still to some extent.

The Talisman mine long employed a rudimentary contrivance in which 1 t. of wood fuel would dry only $2\frac{1}{4}$ t. of ore; this was subsequently replaced by a rotary cylinder consuming but 1 to 3 t. of wood for 15 t. of ore, effecting great economy.

At Waihi, both the structure of the kiln and the method of operating struck the author (Dec. 1897) as being crude in the extreme, and very costly and imperfect. It resembled almost exactly the description given by J. McConnell * as follows:—The kilns are made by excavating a circular pit on the side of a hill, tapering towards the bottom, with a discharge door into

* Trans. Inst. M. & M., vii. 26 (1898).

a tunnel, through which the quartz is passed when dry, and trucked to the breakers. These kilns are charged with alternate layers of firewood and ore. The bottom layer of firewood is lighted, and in time it lights the other layers and more or less thoroughly dries the ore between. As the ore is dried, it is gradually withdrawn from the bottom, and more ore and fuel are filled in at the top, so that the process of drying, once commenced, is continuous. The quantity of firewood used is 1 t. to 3 or 4 t. of ore. The cost of drying usually ranges from 1*s.* 9*d.* to 2*s.* 3*d.* per ton. At the Waihi mill, in 1897, on 87,670 t., it was 1*s.* 3*d.* per ton, firewood costing 5*s.* 3*d.* per ton; in 1898, on 86,695 t., it was 1*s.* 2·13*d.* per ton, firewood having been obtained at 5*s.* 2*d.* per ton. These figures are taken from the official reports. On the other hand, A. H. Bromly * gives the cost at 51·6 cents. per long ton, which is equivalent to 1*s.* 11*d.* per ton.

This method of drying, for some classes of ore, does well enough as a "make-shift," but, as compared with other methods, it is crude, expensive, and unsatisfactory. With ores containing even a small percentage of sulphides, difficulties are created that interfere with the after treatment. The heat is necessarily most uneven, so that portions of the ore are scarcely dried, while other portions are heated to excess. The degree of heat and its consequent effects "depend entirely on the dampness of the ore, and the quantity and nature of the wood charged with it." In this way "the sulphides are decomposed, and varying complex reactions take place, with the formation of free sulphuric acid and sulphates and oxides of iron, which are decidedly objectionable" (McConnell). Another serious difficulty is the formation, in the kilns, of charcoal which is discharged and crushed up with the ore; this precipitates during treatment a portion of the gold taken up by the cyanide solution, and thus causes an actual loss, though what that loss may amount to does not seem to have been a subject of study by New Zealand millmen.

But it has in Queensland, and a recent communication by M. Willgerodt, to the Charters Towers Mining Institute,

* En. & Min. Jl., Nov. 12, 1898.

deserves attention. In treating some sludge from the Brilliant Extended mine, for the St. Andrew's cyanide works (a custom works), very low extractions were got, which led to investigations. When assaying the residues, they were found to contain a considerable amount of charcoal, which, being collected, assayed 19 to 20 oz. gold per ton. A bigger parcel of charcoal was picked out by hand, and after being incinerated and smelted, yielded enough gold to pay expenses. The percentage of charcoal contained in the sludge was not exactly determined, but a similar parcel contained 27 gr. in 8000, or about ·3%; and "1 ton of sludge containing ·3% charcoal might retain as much as 20 oz., divided through 300, equal to 1·33 dwt. gold per ton of ore." How much additional gold the "very fine charcoal, which passed through the screens (90 to 160 mesh per sq. in.) might contain, is difficult to say; perhaps as much, or even more." Repeated assays proved that the "gold is not contained in the charcoal of the sludge before being treated with cyanide solution," but has been precipitated in it. A direct experiment was made as follows:— A sample of sludge containing 14·58 dwt. gold per ton, was, after being air-dried, screened through a sieve (90 mesh per sq. in.), which would allow only small particles of charcoal to pass through; the sample was divided into two parts, and both were extracted in exactly the same way by cyanide solutions, the only difference being that half the weight of the charcoal remaining on the sieve was put back into one of the samples. After a 3 days' treatment, the extraction of sample free from coarse charcoal was 12·72 dwt.; of sample containing coarse charcoal, 11·08 dwt. This means an actual loss of gold equal to 6*s.* 6*d.* per ton. The source of the charcoal is found in this case to be the wood ashes used in grinding concentrates (pan amalgamation), the residues from which operation ultimately joined the battery tailings; but obviously its action will be the same whatever the manner in which its introduction has been brought about.

Another method of drying, advocated on the score of its "simplicity and economy," is to spread the ore on iron plates heated by waste furnace gases or exhaust steam. Apart from

the difficulty of so arranging the mill as to make it fit in with this commendable scheme for utilising waste heat, the item of labour for spreading and collecting the ore has proved in practice to be much in excess of the saving effected in fuel, and the plan has been almost everywhere superseded. Nevertheless, it has the advantage that the drying proceeds in the open air, and in full view, so that complete control can be exercised. And where it is feasible to arrange the flue carrying the furnace gases upon sloping ground, so that the wet ore can be dumped upon an inclined plane, down which it will effect its own passage by gravitation, and especially when this can be accomplished without undue transport from or to the drier and the preceding and succeeding stages, no cheaper or more efficient installation can be desired.

At the modern (1897) chlorination mill at Colorado City, a mechanical drying furnace 100 by 14 ft. is employed, the ore being conveyed from the "beds" in wheelbarrows to the charging hopper, while the drier discharges on to a cable conveyor that delivers to the roll house.

Choice of situation for the drier must in some measure depend upon the stage at which the drying is to be done. It is a common experience that the ore drawn from different parts of a mine, or at different times, will show wide variations in moisture contents. Often, also, there is considerable range according to the size of the material, the smaller always being the wetter; and while that portion which passes through the grizzlies (p. 10) is pretty certain to require drying, it may be that this process can often be dispensed with in the case of the large lumps which have to traverse the breaker, and particularly if breaking is carried to a rational point (½ to 1 in.) before the fine-crushing machine is entered. Provision must be made accordingly, allowing such ore as does not need to be dried to pass to the next stage without wasting fuel on it, or entailing additional labour instead.

*Drying Machines.*—Machines for drying ore are of two principal kinds, rotary driers and shelf kilns. Both are automatic and continuous in operation, and in both the ore comes into actual contact with the heated gases. They differ only

in this—that the former are almost horizontal, and effect the exposure of the ore, and its transmission from the feed to the delivery end, by a rotary motion; while the latter are vertical, and the ore passes through them by gravity. The rotary drier is the more popular for several reasons; it is less costly to build, and more easily fed, and though it is a little more expensive in operation, by reason of the power required for its rotation, the work it does is much better controlled, and the duration of the drying can be hastened or retarded at will. With the shelf kiln, the ore must be elevated in order to fall through it, and there is no way of regulating the speed of passage. In both, previous breaking of the ore is desirable to minimise wear and tear.

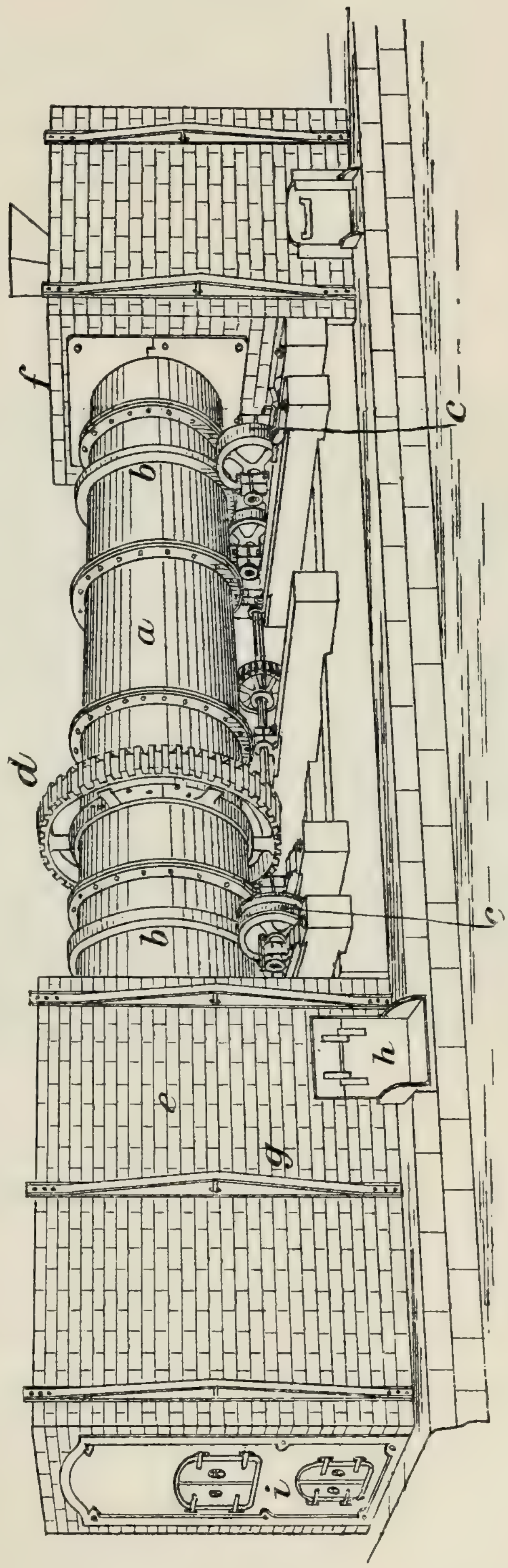



Fig. 82.—Rotary Drier.

The rotary drier, as made by Fraser & Chalmers and the Gates Iron Works, is shown in Fig. 82. It consists of a cast-iron cylinder *a*, in several sections for convenience of handling and transport, provided with 2 tracks or tyres *b*, on which it rotates, supported by rollers *c* underneath. Motion is transmitted through pulleys and the gear-wheel *d*. The cylinder is of larger diameter at the fire end *e*, and ore from

the breaker is fed in at the smaller end *f*. The cylinder's axis is placed horizontally, but owing to its conical form, the ore travels gradually towards the fire at the larger end. Shelves or wings, arranged spirally inside, raise the ore and shower it through the flames. The dried ore is dropped immediately into a pit *g*, from which it is drawn at the cast-iron door *h*, and by means of shoots lined with sheet iron, is conveyed to the fine crushers. The fireplace is at *i*. The usual size is 44 in. diam. at the large end, 36 in. at the small, and 18 ft. long ; total weight of ironwork, about 9½ t. ; capacity, 30 to

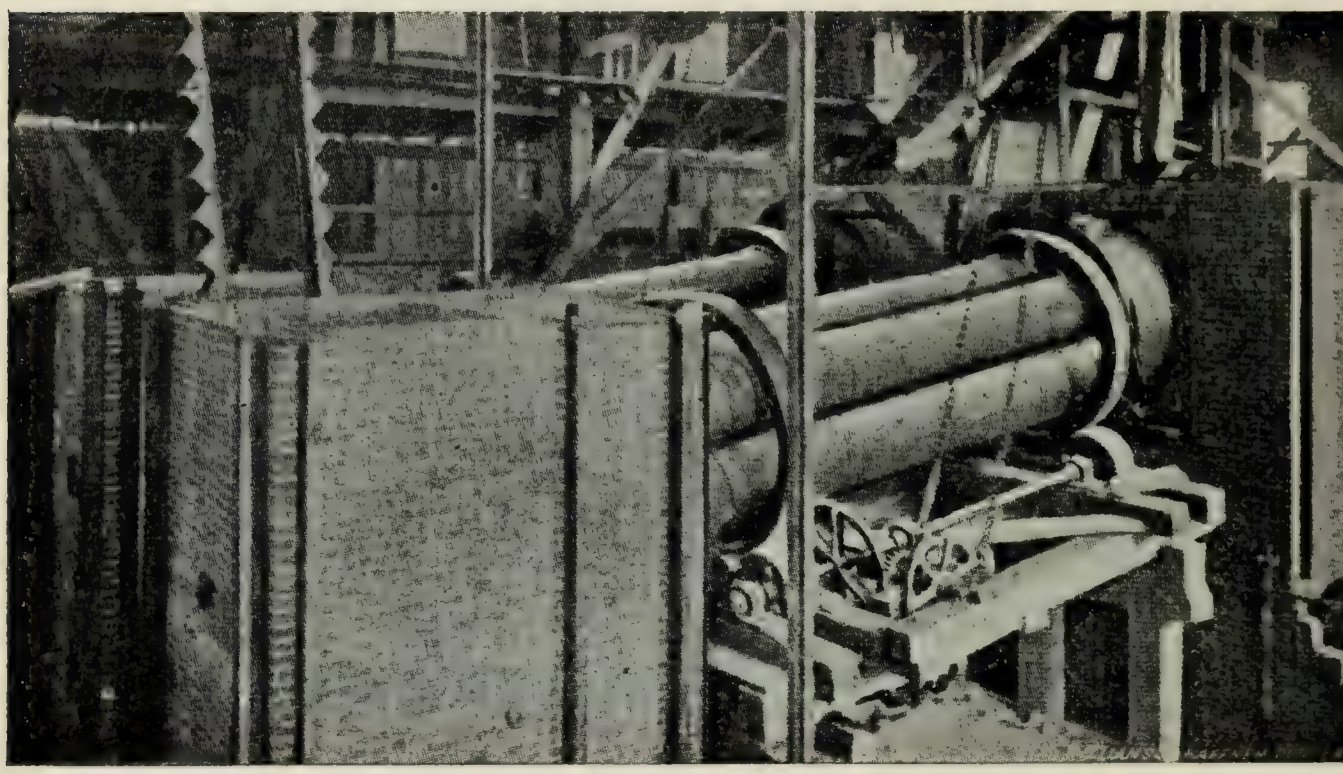


FIG. 83.—ARGALL DRIER.

40 t. per 24 hours ; price, about 350*l*. It requires also 1100 fire-bricks for lining. A chimney of some kind must be provided, or connection be made with an existing one.

The Argall drier, as made by F. M. Davis, Denver, and shown in Fig. 83, consists of a number of fire-brick lined steel tubes nested together inside two steel tyres, resting on and driven by steel-faced carrying rolls. These latter have bearings of the ball-and-socket type, lined with phosphor bronze, and are supported by cast-iron sole plates resting on a heavy framework. This framework is adjustable, so as to accommodate the drier to the needs of the ore under treatment,

making the passage slower or quicker as required. The fire-box is of brick. Two sizes are made, having the following dimensions :—

| | | |
|---|---|---|
| Capacity per 24 hours | 70 to 100 t. | 180 to 200 t. |
| Weight of ironwork | 18,000 lb. | 30,000 lb. |
| Length of tubes | 17 ft. | 25 ft. |
| Diam. of tubes inside lining | 20 in. | 25 in. |
| Weight of fire brick | 4 t. | 6 t. |
| Number of fire bricks | 2000 | 3000 |
| Number of common bricks | 8000 | 10,000 |
| H.P. required | 1 | 1 |
| Price of ironwork | 360*l.* | 480*l.* |

A typical shelf kiln is the Stetefeldt,* following in principle the Hasenclever dust-burning pyrites kiln used in the sulphuric acid trade. The shelves are inclined and zigzag, with openings or slits where they meet, on which the ore rests in a stratum governed by the width of the slits and the pitch of the shelves. As a charge is withdrawn at the bottom, so the whole contents is let down a stage. Hot gases are admitted into each triangular space by flues from alternate sides of a hot-air passage. The fire is at the base, but the hot air is made to pass downwards through the kiln, so that it is hottest when it first strikes the ore, and evaporated moisture is not condensed by meeting cold wet ore. There is little loss by radiation, and the exposure is longer than in a rotary drier, hence it is claimed less fuel is consumed. No power is required, and no dust is created. The iron part of the kiln weighs about 10½ t., and for the structure are used 30,000 bricks. It is patented.

A sort of shelf kiln, erected on an incline, as shown in Fig. 84, was introduced by W. Hutchison at the Crown mill, Karangahake, N.Z. The ore fed in at *a* slides over a succession of fire-clay slabs or iron plates *b*, built into the side walls of the furnace, and overlapping in a way that prevents the ore escaping into the hot-air passages *c*. The back wall *d* may be lodged against a hillside, or be built on a rubble filling supported by another wall behind. The front wall *e* is iron plate covered with a bedding of non-conducting material, such

* Trans. Am. Inst. Min. Engs., xii. 95.

as sand, or, in this case, pumice-dust. A revolving door at *f* controls discharge into shoot *g*. A furnace to dry 4 t. at a charge requires 15,000 bricks and costs 500 to 600*l*. It con-

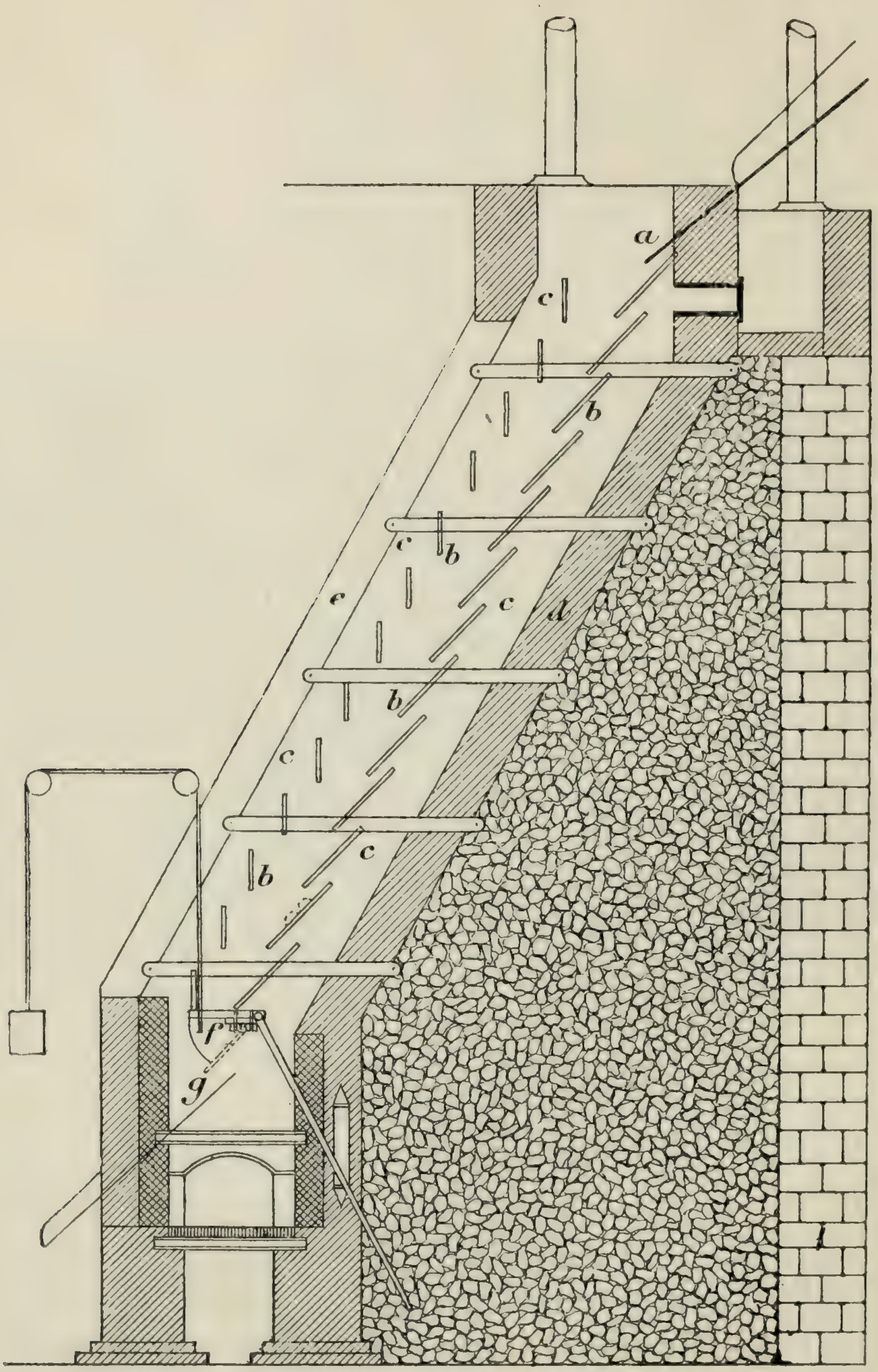



FIG. 84.—SHELF DRYING FURNACE.

sumes 1 t. cordwood for every 8 t. dried. The door *f* encounters a fierce heat, and is soon burned or warped. The access of hot air to the ore is very partial, and there must be

considerable condensation of moisture where the ascending saturated air meets the incoming cold wet ore.

A desirable condition with all mechanical driers is a regular and constant feed. The Gates Iron Works have introduced a simple and efficient appliance for this purpose, as shown in Fig. 85. The hopper *a* is of sheet steel, attached to a substantial cast-iron frame *b*. Below the hopper, a tray *c* is suspended by iron straps *d*, which can be adjusted to give

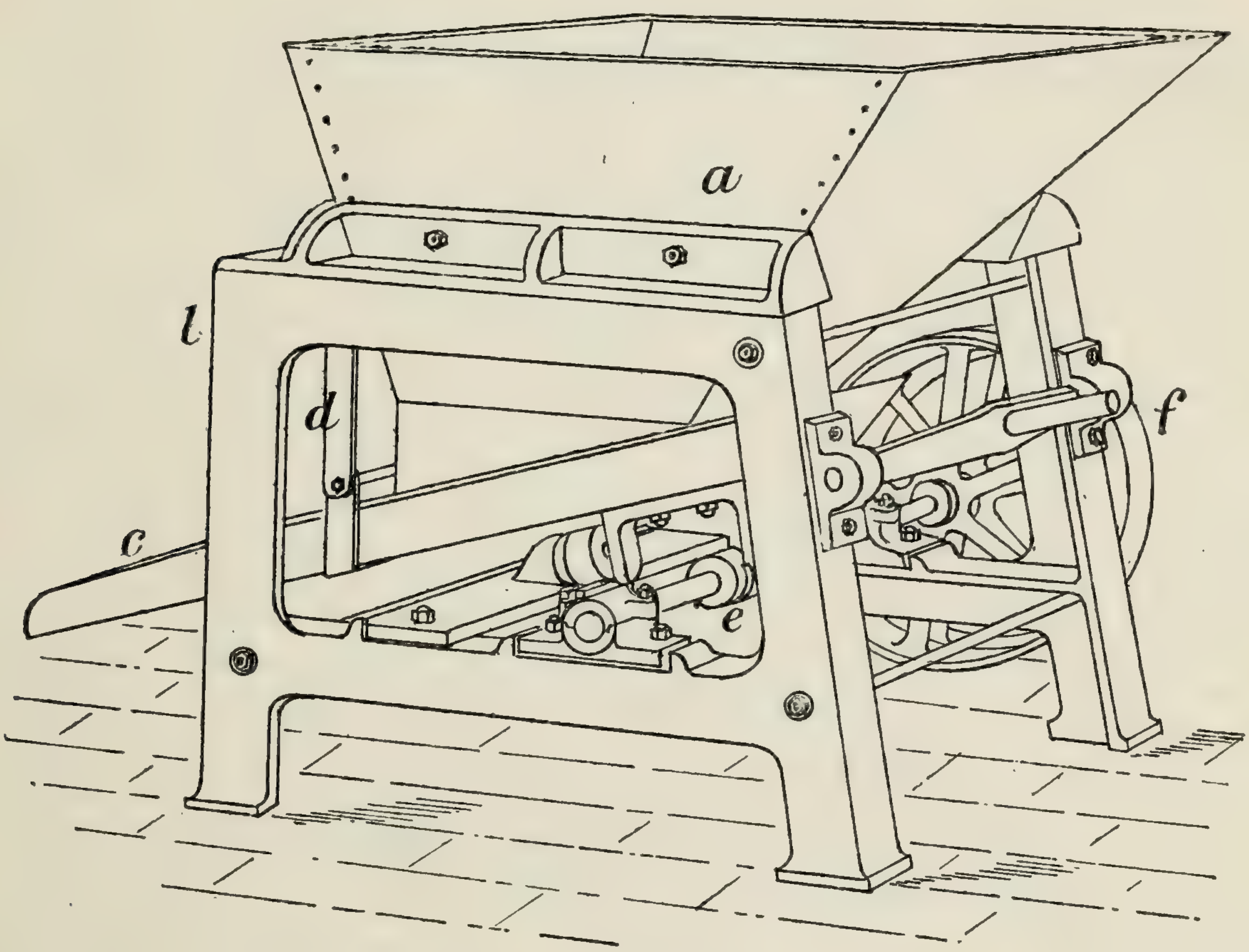



FIG. 85.—FEEDER FOR DRIERS.

the tray the degree of inclination required by the ore. Motion is imparted to the tray by a cam *e* working against a steel face; upon each revolution of the cam, the tray is liberated, and returns with force against a rubber cushion, being assisted in this movement by a spring. The resultant percussion drives a portion of the ore over the end of the tray. The fast and loose pulleys *f* are calculated to make 50 to 70 rev. per minute. The feeder weighs 800 lb., and costs about 30*l*.

The cost of mechanical drying must necessarily depend to some extent upon the wetness of the ore, but still more upon

the efficiency of the automatic arrangements for feed and delivery, labour being as a rule a much more costly item than fuel. Under average conditions, it is generally somewhere about 6*d.* per ton, that is, of course, where simple drying and no calcination is effected.

**Dry-Milling Machinery.**—In order of their importance, dry-milling machines are principally rolls, ball mills, and stamps, with some few modern inventions which may be grouped under the head "Sundry." It is assumed, of course, that the rational practice is followed, in dry as in wet milling, of passing the ore first through a breaker, or better still, two breakers, so as to reduce it to a size at which the fine-crushing machinery can be most effective. This principle of graduated reduction cannot be too often or too loudly insisted upon. It is, in fact, the very foundation of economy in milling, each type of machine having a certain range of efficiency, with definite limits, outside which it ceases to do good work. The use of coarse and fine breakers in succession has been already discussed (p. 41); the same argument applies here. Two, or even three sets of rolls may well be made to deliver their product from one to the other, and, where very fine crushing is desired, the ball mill is most appropriate for the last stage. In urging this system of graduating the work of the mill, it is not forgotten that many small undertakings could neither bear the cost of so much machinery, nor keep it employed when erected, so that the reverse of economy would be experienced. The remedy for this is fewer small mills and more large ones, the small mine contenting itself with mining, and sending its produce to a custom mill.

*Rolls.*—For intermediately fine dry crushing, fed say with $\frac{1}{2}$-in. material and delivering it at $\frac{1}{8}$ to $\frac{1}{16}$ in., rolls are a thoroughly satisfactory form of machinery, particularly if a minimum of dust in the product is desirable.

The old Cornish rolls have given place to a variety of improved patterns, prominent among which are the so-called Krom rolls, made in this country by Bowes Scott & Western, Westminster.

Krom rolls consist essentially of two sets of cast-iron rolls *a*, on steel axles *b*, carrying steel tyres *c* (see Figs. 86, 87). The rolls range generally between 26 and 30 in. diam.,

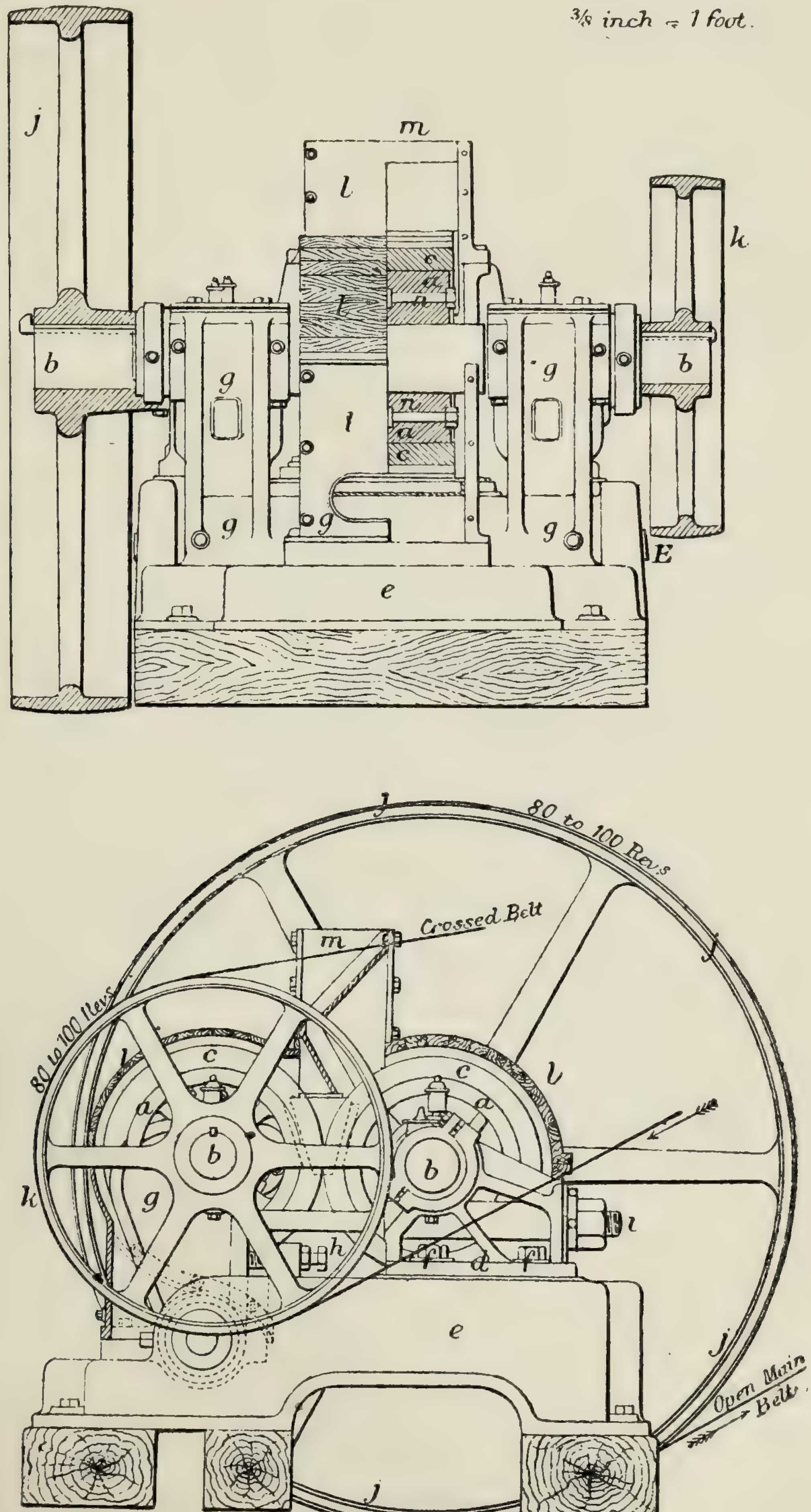



FIG. 86.—KROM CRUSHING ROLLS.

including the steel tyre. The tyres are made of the best open-hearth steel, and are 2½ in. thick on 26-in. rolls, and 2¾ in. thick on 30-in. rolls, and can be worn down to ½ in. with safety. They have been worn as low as ¼ in., but there is danger that they will then spring and become loose. The pillow-block *d* of one of these rolls is firmly bolted to bed-plate *e* by nuts *f*. The second roll is set in a swinging

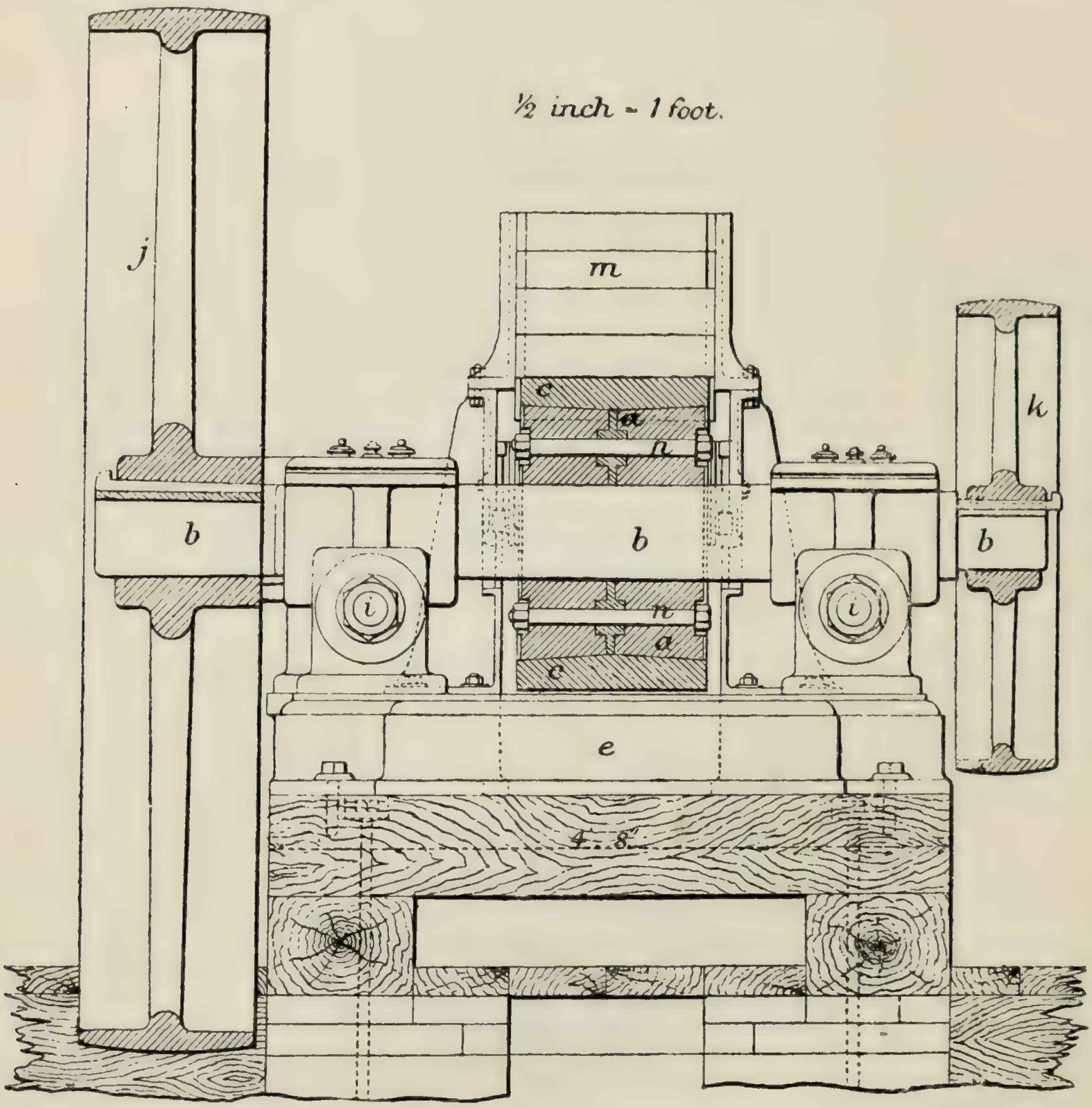



FIG. 87.—KROM CRUSHING ROLLS.

pillow-block, fixed in two strong reinforced cranks *g*, which rotate in a journal *h*, set in the cast-iron frame *e*. These double cranks are of exactly the same width as the roll. The shaft which supports them is 11 in. diam. Their journals are therefore pivoted with those of the rolls, and the two are thus forced to move together, so that this roll and shaft are always parallel to the other roll. The distance between the two

rolls is regulated by screws *h*, one on each side, with jam-nuts to prevent any motion after they are once set. The rolls are held to their position by two heavy bolts *i*, also with jam-nuts, so that the position being once fixed, the distance of the surfaces cannot be changed by any action of the machine but the wear of the rolls. Formerly the distance was fixed by having the pillow-boxes slide on the bedplate, but this was found not to work well, as they constantly became loose and thus pounded on the bed. It was also found almost impossible to get the surfaces of the two rolls parallel, and they wore unevenly. To the axis of the fixed roll is keyed a large driving wheel *j*, 7 ft. diam. and 15 in. wide; and to the movable one, a smaller wheel *k*, 50 in. diam. and 8 in. wide. The rolls are covered with housing *l*, to which an exhaust fan is attached, so that no dust escapes into the air; this protects the journals of the machine from abrasion. This housing carries a feed box *m*, arranged with a series of inclines so as to spread the ore in a continuous and even sheet between the surfaces of the rolls. Magnets are provided in the feed-box to catch any pieces of steel which may be in the ore from broken tools, and which might indent the surface of the rolls.

The whole machine is self-contained and compact, a pair of 26-in. rolls occupying a ground space of only 7 ft. by 7½ ft. The tyres (which for the 26-in. roll weigh 816 lb. each) are held in place by two cast-iron heads, which are slightly conical in shape. One of these is shrunk on to the shaft and permanently fixed, the other is slit on one side and slips on to it. Both the heads are so placed on the shaft that the smaller diameter will be towards the centre. The steel tyre is turned out on the inside to correspond to this, so that it can be easily slipped over the permanent head and the loose one brought up to it. The two are securely fastened together by bolts *n*, so that when the movable head is drawn up to the permanen one, the slit in it closes, and makes it perfectly tight on the axle. It has been found expensive, on account of turning the inside of the tyre, to have it all made in one piece, so that it is sometimes cast in two pieces, which are turned to fit

together in the centre. The bolts *n* draw them together perfectly tight. It was thought at first that the steel would wear unevenly on the two parts of the tyres thus joined, but it does not, and the economy and convenience realised by the construction are great.

When the tyres are worn so thin as to be in danger of springing, they can be very easily removed and others substituted for them. It is a good plan to have extra hubs with the tyres on, so that there may be no delay when the tyres are worn out. The latter can then be replaced at leisure. In order to provide for abrasion on the sides of the machine, cheek-pieces of composition metal are provided, which can be easily taken out and replaced when too much worn. The putting on of a new set of tyres requires but a short time, so that the delay when the rolls are to be replaced need be very slight, more especially if extra sets of rolls are kept on hand with the tyres already shrunk on the hubs, so that they may be adapted at once.

Rolls are generally arranged in double sets, one of which receives the coarse ore from the crusher and delivers it on to a screen of a determined size. That portion which does not pass through the screen is carried back to the first rolls. What passes the screen falls upon the second pair and from thence to another screen. Part passes this last screen and goes on for treatment ; part remains and goes back to the second rolls again. Whenever the screens indicate wear in the rolls, the latter are stopped and reset. When finer crushing is required, three pairs of rolls are used ; while for small outputs, or where a very coarse product is needed, the machine may be used as a single crusher.

Krom's rolls are made of three sizes, with varying lengths and diameters. These are 22, 26, and 30 in. diam., and 14, 15, and 16 in. long. There is no practical advantage in making them longer, while the difficulty of fitting and making the tyres would be very much increased.

Power was formerly applied to rolls by gearing, which is liable to get out of order, and limits the speed at which it is possible to run. Pulleys are now used, a belt passing over

both large and small wheels. This allows of attaining a speed of 80 to 150 rev. a minute, or higher if desirable, which was impracticable with gear. The capacity of the rolls is thus increased, while the wear is confined almost entirely to the crushing faces. It does away with the noise of the gearing, and reduces the risk of breakage. Both rolls, when the machine is in operation, travel with the same speed of surface, but the small pulley is so speeded that when there is no ore between the rolls, it travels 1 or 2 rev. per minute faster than the large one. The reason for adopting this double system is that one roll being movable and the other stationary, it would not be good construction to place a large pulley on a movable pillow-block. When ore is between the rolls, a single driven roll will cause the other to revolve, so that if most of the power to do the work is applied to the large wheel, only a small part of it will be necessary to ensure that the other roll will always bite, when fed with ore, and be kept in motion when the ore at any instant fails. Most of the strain of the rolls is taken upon the bolts *i*; and, by the use of two bolts *i* and *h*, with nuts *f*, all of which are provided with double nuts, no pounding action is possible.

A number of American firms turn out crushing rolls more or less closely allied to the Krom system. That there are very wide differences in quality, however, is shown by the fact that some are catalogued at just half the price of others. The most responsible makers are Fraser & Chalmers, the Gates Iron Works, and the E. P. Allis Co. The hardened steel tyres of the English makers are much more durable than the forged steel sometimes furnished, and do not lose more than ·1 lb. steel per ton milled to 40 mesh.

Fraser & Chalmers have introduced "high-speed" rolls in two standard sizes :—56 in. diam., 8 in. face, 160 rev. a minute, 24,700 lb. ; 44 in. diam., 5 in. face, 180 rev. a minute, 15,385 lb. They have heavy shafts with large journal surface, two fly-wheels on the main driving shaft, and adjustment for keeping rolls in line. The housing for preventing escape of dust may be conveniently opened for examination. At the shaft openings are dust-plates set up by springs, and there is a

special device for keeping dust from the bearings. A test of 56 × 8-in. rolls showed that they could easily crush 300 t. of hard granite in 24 hours, with material fed at 2-in. ring and rolls set $\frac{1}{4}$ in. apart. Of the product in this case, 10 % ran $\frac{1}{2}$-in., 90 % went through $\frac{1}{2}$-in. screen, 60 % through $\frac{1}{4}$-in., 37$\frac{1}{2}$ % through $\frac{1}{8}$-in., 27 % through $\frac{1}{16}$-in., and 12 % through 30-mesh.

It would seem that different ores require different speeds for maximum capacity, which indeed is only common sense. While 900 ft. a minute is an average peripheral speed giving good results, at the Golden Gate mill, Mercur, Utah, 700 ft. gives a greater efficiency, the crushing being from 1$\frac{1}{2}$-in. feed to $\frac{1}{4}$-in. delivery.

The overlapping edges of the roller shells, being exempt from the wear and attrition borne by the rest of the shell surface, are gradually transformed into flanges, which tend to increase the side thrust and lessen the capacity of the machine.

When used for fine crushing, the shims may be removed from the tension bars, thus allowing the shell surfaces to come together.

If correctly adjusted, the small beads that appear in wear at the ends of the shells should be of the same size. When this bead has elongated the shell surfaces sufficiently to cause the ends to wear against the face of the "cheek" or wearing plate, it should be removed by lifting the curtain at end of the housing, and applying an ordinary hardened-face blacksmiths' set-hammer on the head, holding the hammer at a slight angle so as to cause a shearing effect.

The nuts at the ends of the bolts holding the shells to the hubs should be kept tight; the shells, particularly if worn, will sometimes expand, causing them to work loose on the hubs.

In adjusting the spring pressure, it is of vital importance to provide an equal amount on each side, as a difference in this respect will cause an uneven wear on the shells.

To obtain the full capacity of a set of rolls (especially in fine crushing), it is absolutely necessary that the faces be kept true; if through long service they become concaved, or worn

into grooves, they should be turned off. A lathe for this purpose, complete with mandrel, etc., should be part of the mill equipment. Bowes Scott & Western furnish a special emery grinder.

Another requisite for satisfactory working, to obtain full capacity and ensure long life, is proper feeding. The material should be regularly and evenly distributed across the entire width.

An automatic feeder, which can be adjusted to any rate of feed, is made by the Gates Iron Works, and shown in Fig. 88.

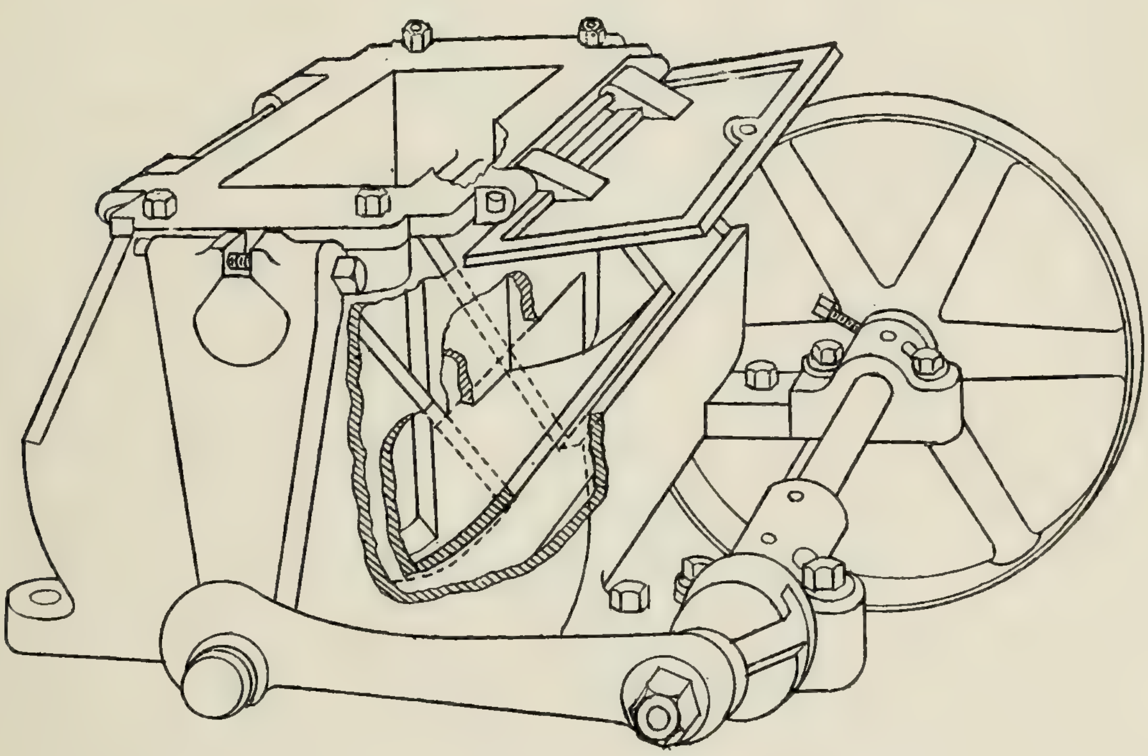


FIG. 88.—FEEDER FOR ROLLS.

The larger size, weighing 300 lb., costs 12*l.* The starting and stopping of the feeder are simultaneous with those of the rolls, thereby preventing them from running light, or being overfed with material, by slipping or breaking of belts, or other causes. The ore upon entering the feeder is divided by a central partition, and falls on either side directly on a curved bottom plate. An oscillating motion is imparted to this plate by a pitman connected to an adjustable crank-disc driven by a belt from the roll-shaft. The central partition being stationary, the material is held while the bottom plate moves from under a portion, dropping it directly between the rollers. Being double-acting, a practically continuous flow of material is

maintained. The pitman can be readily adjusted on the disc to any position, thus giving any required swing to the bottom plate of the feeder, and keeping the flow of ore at all times under perfect control. It is equally serviceable for either fine pulp or coarse ore.

A few typical instances may be given of the work being done by rolls.

At the Mercur mill, Utah, the ore is first broken to 1 in.; from the breaker, it passes to 20-in. rolls which reduce it to $\frac{1}{2}$ in.; thence over a grizzly and to second rolls which reduce it to $\frac{1}{4}$ in. These sizes are maximum; 61 % remains on a No. 12 screen, and 26 % passes a No. 30.

In a Black Hills, Dakota, plant, the ore is broken to $1\frac{1}{2}$ in. max., but a large portion will pass a $\frac{3}{4}$-in. screen. An elevator carries the crushed ore to a feeder, and thence to a revolving drier. From the drier, the ore passes to a 36 in. × 7-ft. revolving screen. The rejections from the screen are reduced to $\frac{3}{4}$ in. by a No. 3 Gates crusher. The ore passing through the screen, as well as the product of the crusher, is carried by an elevator to a bin over the 36 × 16-in. roughing rolls. The contents of the bin are spouted to the rolls feeder. The product of the rolls is elevated to a screen with $\frac{1}{4}$-in. mesh openings, the screen rejections passing back into the bin; that portion passing the screen is delivered to a bin supplying 3 sets of 26 × 16-in. rolls. The product of these rolls is elevated and passed over inclined screens made of wire cloth, and forming a spout, which is given a slight but rapid jar by suitable mechanism, and affords a product equal to a 10-mesh revolving screen, but with the advantage that it never clogs, and has large capacity. The rejections from the screens are spouted to a third set of 26 × 16-in. rolls for finishing.

The crushing plant in the modern (1897) chlorination mill at Colorado City, Colorado, occupies a building 92 × 32 ft., and consists of 6 sets of 30 × 14-in. Allis Co.'s belt rolls, with positive feed attachments, 6 revolving screens (12 mesh with No. 19 wire), 6 belt and bucket elevators, fitted with a feed hopper, with screw feeder driven from lower shaft in elevator. Placing a hopper and positive feeder at the foot

of the elevator assures a constant and uniform flow of ore; consequently, neither elevator, screens, nor rolls can become overloaded. The main line-shaft is on the ground floor, directly under the rolls. This does away with all overhead belts in the roll-room proper, except those driving the elevators, which are comparatively small, and high enough to be out of the way. The system of crushing followed is gradual reduction, although each set of rolls with one elevator and one screen constitutes an independent system. The flow of the ore is as follows: From discharge of drier, the ore is trammed in bottom-discharge cars to the elevator feed-hoppers. A screw in the bottom of the hopper feeds the ore to the elevator and discharges into the screen. The coarse material goes to the rolls, and, after crushing, is spouted to the elevator feed-hoppers, mixed with the supply coming from the drier, and elevated again to the screen and rolls. Thus each set of rolls with its elevator and screen constitutes an independent crushing system, which can be stopped or started without interfering with any other system. A belt elevator returns the finished pulp to the roaster house. The plant was especially designed to treat the ores of Cripple Creek and other districts that are suitable for chlorination. All ores are received from railroad cars at the sampling mill, where they are sampled before bedding or storing, being unloaded by hand labour with wheel-barrows, and dumped directly into the crusher. From the crusher the ore is elevated to two revolving screens, which are covered with wire cloth of $\frac{3}{4}$ mesh. What passes through these screens drops through a spout on the shovelling or sampling plate. The reject from the screens is returned to crushing rolls, and, after crushing, is returned to the screens by the same elevator. In this way all the ore is reduced to $\frac{3}{4}$ in. and less in size before going to the sampling plate. As the ore drops from the spout on to the plate, it is shovelled into pockets or charging bins. Every tenth shovelful is thrown aside for a sample. This sample is then cut down on quartering plates and treated in the usual manner. The effective crushing duty when running 5 sets of rolls (one being in reserve) is 230 to 250 t. per 24 hours.*

* H. V. Croll, En. & Min. Jl., Oct. 8, 1898.

At the recently built Delano mill, Colorado,* also for chlorinating telluride ores, the material is delivered to a 9 by 15-in. Blake crusher, elevated to a set of coarse rolls 16 by 36-in., set to ¼-in. The ore leaving the rolls slides along a chute, in which a horizontally moving plate is set, provided with a slot one-tenth of the width of the chute. As the ore passes over the plate, one-tenth of it is delivered on the floor as a sample, the balance going to the bedding floor. The sample thus obtained is split-shovelled, recrushed and again cut down to a suitable size for the sample room. Here it is ground in a "coffee mill," finishing on a bucking board until all will pass a 120-mesh sieve. On most of the ores this degree of fineness seems necessary to get results that will closely check. A bedding floor is found to be indispensable for mixing the widely different ores. The bedded ore is elevated to storage bins, passes thence through a drier, and is elevated to 16-mesh screens, the oversize being returned to 2 sets of fine crushing rolls 30 × 14 in., the discharge from the rolls falling into the same elevator as the dried ore, and thence to trommels. The latter are placed over bins of 25 t. capacity each. The speed of the coarse rolls is 17 rev. per min., and of the fine rolls 30 rev., this slow speed varying as the diameters give a circumferential speed nearly equal to the speed of the falling ore ; the result of this is that there is no dragging or grinding of the ore, but simply a cracking, which is regarded of high importance in hindering production of slimes. Mesh determinations on average ore crushed to 16 mesh (with heavy rolled wire, almost equal to 20 mesh) showed as follows :—On 20, 3·7 % ; on 30, 34 % ; on 40, 16 % ; on 60, 16 % ; on 80, 9 % ; on 100, 6·2 % ; through 100, 15·1 %. The belting used throughout for elevators is "leviathan" cotton duck, which has proved vastly superior to rubber for conveyor purposes. Water sprinklers and exhaust fans are used everywhere for minimising the dust nuisance.

Two sets of Gates rolls (Nos. 1 and 2) running for 72 days and dealing with over 20,000 t. of Rand banket, at the rate of 8 ¾t. per hour (not their full capacity), required 43 i.h.p.,

* C.C. Burger, En. & Min. Jl., Jan. 29, 1898.

No. 1 (reducing $\frac{1}{2}$ in. to $\frac{1}{4}$ in.) taking 20, and No. 2 (finishing) 23. The character of their work is shown below.*

ROLL No. 1, COARSE ($\frac{1}{2}$ in. to $\frac{1}{4}$ in.).

| Free Milling Ore. | | | Pyritic Ore, $\frac{1}{14}$ in. | | |
|---|---|---|---|---|---|
| Size of Mesh. | Ore. | | Size of Mesh. | Ore. | |
| | Entering. | Leaving. | | Entering. | Leaving. |
| in. | % | % | in. | % | % |
| $+\frac{1}{4}$ | 72·7 | 19·6 | $+\frac{1}{4}$ | 60·4 | 10·5 |
| $+\frac{1}{8}$ | 6·9 | 9·4 | $+\frac{1}{8}$ | 2·6 | 10·5 |
| $+\frac{1}{16}$ | 7·9 | 29·4 } | $+\frac{1}{16}$ | 15·5 | 31·2 |
| $-\frac{1}{16}$ | 12·5 | 44·6 } | $+\frac{1}{20}$ | 12·9 | 18·7 } |
| | | | $-\frac{1}{20}$ | 8·6 | 29·1 } |

ROLL No. 2, FINISHING (from $\frac{1}{4}$ in.).

| Free Milling Ore. | | | Pyritic Ore, $\frac{1}{14}$ in. | | |
|---|---|---|---|---|---|
| Size of Mesh. | Ore. | | Size of Mesh. | Ore. | |
| | Entering. | Leaving. | | Entering. | Leaving. |
| in. | % | % | in. | % | % |
| $+\frac{1}{8}$ | 32·0 | 0·4 | $+\frac{1}{8}$ | 22·5 | 0·9 |
| $+\frac{1}{16}$ | 42·7 | 47·3 } | $+\frac{1}{16}$ | 40·0 | 21·3 |
| $-\frac{1}{16}$ | 25·3 | 52·3 } | $+\frac{1}{20}$ | 27·5 | 37·7 } |
| | | | $-\frac{1}{20}$ | 10·0 | 40·1 } |

*Note.*—The bracketed quantities represent finished product.

The weight of material worn out (7325 lb.) and discarded as useless (3281 lb.), totalled 10,606 lb., and cost 3·536*d.* per ton milled.

The "centrifugal" rolls introduced by the Sturtevant Mill Co., Boston, U.S., crush by utilising the centrifugal force generated by their rotation. They thus dispense with springs for keeping the roll faces up to their work. They also automatically balance themselves, and thus are able, it is said,

* F. White, Trans. Inst. M. & M., vii. 134 (1899).

to run very steadily, without the need of fly-wheels. No record of any work accomplished by them, however, is yet to be had.

Another innovation is to cushion and sectionise the rolls,*

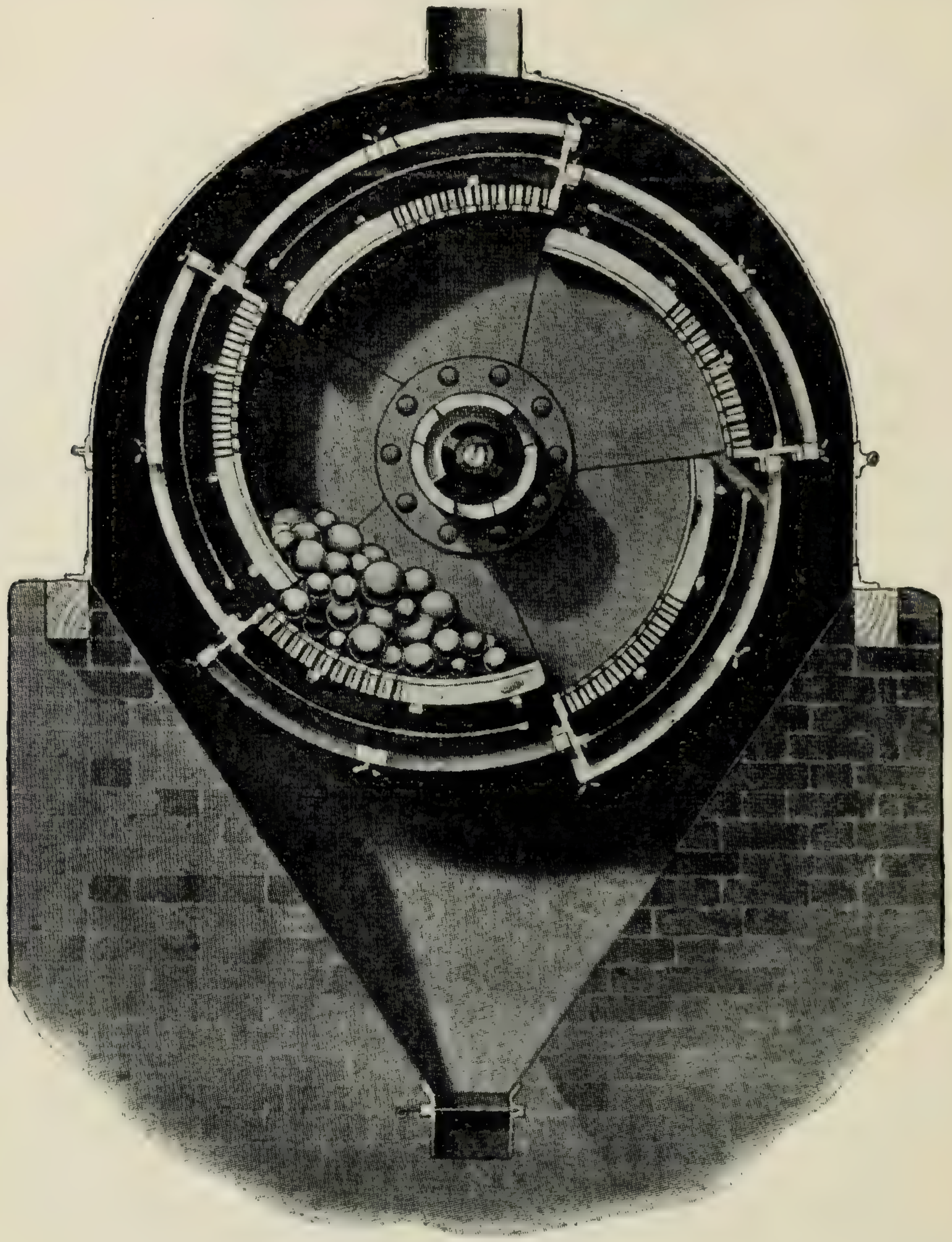


FIG. 89.—BALL MILL.

so as to prevent the finer particles of ore from passing through uncrushed when a large lump forces the rolls apart. This is accomplished by making one of the rolls in several sections or rings about 2 in. wide, only one of these rings being forced back when the hard lump arrives. For each ring, there is a stiff

* J. W. Pinder, Trans. Am. Inst. Min. Engs., xxviii. 243 (1898).

rubber cushion, which makes a snug fit both on the shaft and in the bore of the hub. The cushion is ¾ to 1 in. thick, according to the size of the machine. The sections of the roll have skeleton cores, and are driven by two steel arms in the form of lathe dogs, one on each side. These arms are keyed to the shaft, and each passes through and presses against the spokes of half the sections. If the work of rolls is properly graduated, there would hardly be any occasion for making such provision as this invention contemplates.

*Ball Mills.*—The ball mill, as shown in Fig. 89, consists of a revolving cylinder, into which the ore is automatically fed through the hollow axis. The sides of the cylinder are armour-plated, and the periphery is of the peculiar form indicated, and composed of hard steel curved plates with perforations. It is surrounded by a series of screens. The grinding is done by very hard steel balls of various sizes, inside the cylinder, rolling over and falling on the ore. The machine is best adapted to very fine crushing, and, as common sense would indicate, should be fed with material which has already passed through rolls or other fine crusher. An advantage of the ball mill is that, being tightly closed, practically no dust escapes. The illustration represents the machine as made by Bowes Scott & Western, Westminster, London.

BALL MILLS.

| No. | Dimensions. | Weight. | Weight of balls. | H.P. |
|---|---|---|---|---|
| | ft. in. ft. in. | t. | cwt. | |
| 0 | 2 2 × 1 1 | ½ | 1 | ¼ |
| 01 | 2 11 1 8 | 1 | 2 | ½ |
| 1 | 3 5 2 4 | 2 | 3 | 1½ |
| 2 | 4 4 3 1 | 3½ | 6 | 4 |
| 3 | 5 1 3 3 | 4½ | 9 | 8 |
| 4 | 6 0 3 3 | 6½ | 13½ | 12 |
| 5 | 7 5 3 10 | 9½ | 21½ | 18 |
| 6 | 7 5 4 6 | 10½ | 26½ | 23 |
| 7 | 8 10 3 10 | 11 | 31½ | 26 |
| 8 | 8 10 4 6 | 12 | 39½ | 28 |

At Deloro, Canada, the pulp from ball mills, after classification in pyramidal boxes, gives 60 % of leachable product and 40 % of slimes.

An example of highly efficient work by a ball mill is given by G. M. Barber,* who ground with eight machines of this class some 25,000 t. of abnormally hard material in the Guanaco district of Chili. The wearing parts were made of special malleable cast steel, and wore till only the thinnest shell was left, the mill running day and night for 3 months without any stoppage for repairs. The output of a No. 4 mill was about 6 t. per 24 hours reduced to pass 80 mesh (6400 holes per sq. in.), and the grinding was most uniform. The wear of steel from liners and balls amounts to 5·69 lb. per ton milled, 75 % of which is actually abraded, the rest being in worn-out plates, etc.

But on the score of economical application of power, the ball mill leaves much to be desired. A No. 4 mill demands 12 to 18 actual h.p., which, on the output above given, means 2 to 3 h.p. per ton milled per 24 hours ; whereas 900-lb. stamps at 7-in. drop, 90 per minute, do not require quite 2 h.p. per head, and could be counted on to mill 1½ to 2 t. per head per 24 hours to 80 mesh. However, in this case, economy of power was not the desideratum. Absolute dry pulverisation was sought, and this the ball mill accomplishes better than any other machine. As a granulator, it is not serviceable.

In direct opposition to this opinion, Prof. R. H. Richards† says ball mills are "especially adapted to crushing gold ores where the sliming of tellurides is to be avoided." Yet they are most largely employed in Western Australia, with the distinct object of reducing the telluride ores to an impalpable condition, so that their contained gold may be liberated for bromo-cyaniding without the need of previous roasting.

A No. 5 mill at Pearce, Arizona, is quoted as affording 23 t. per 24 hours through 40 mesh, using 12 h.p.

The Talisman (N. Z.) mill, in the year ending June 30, 1899, passed 4135 t. (out of a total of 11,014 t.) through a

* Trans. Inst. M. & M., v. 102 (1897).

† Mineral Industry, viii. 754 (1900).

ball mill, which gave an average duty of 16·035 t. per 24 hours.

Objections are being urged against the ball mill, in West Australia, that the iron abraded from the wearing parts of the machine is detrimental to the use of bromo-cyaniding methods, which are coming into favour for treating the sulphide ores without roasting.

This has led to the "grit" mill being borrowed from the cement industry in substitution for the steel ball mill; in this the grinding is done by hard, smooth sea pebbles. The machine, to be had of W. Stamm, London, is shown in Fig. 90, and consists essentially of a long wrought-iron drum *a*,

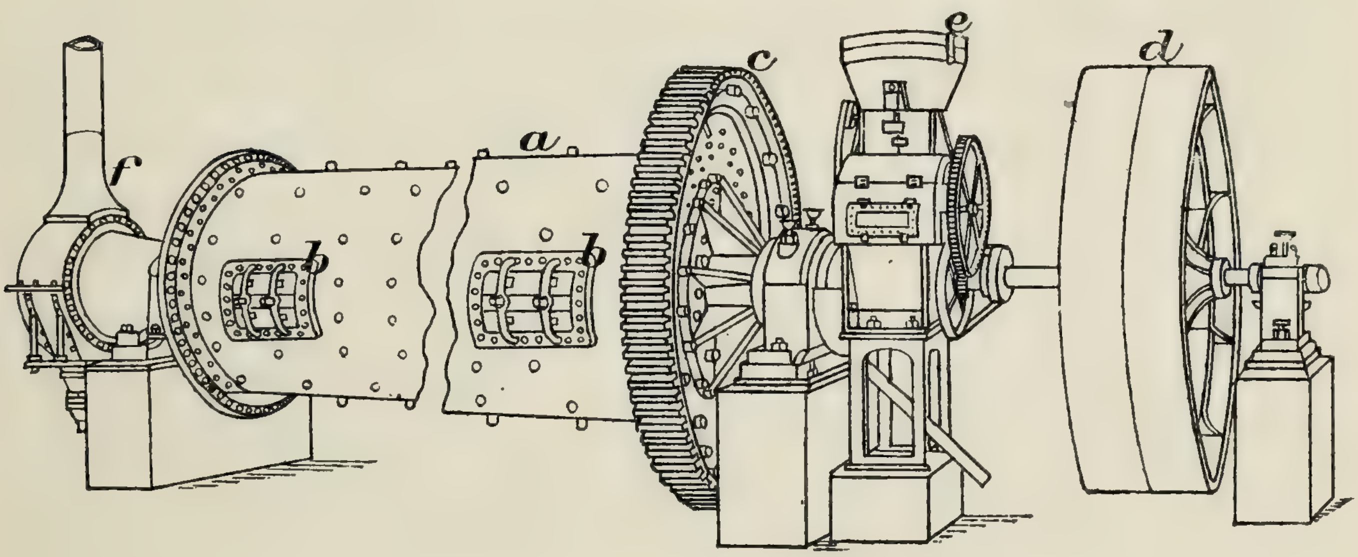



FIG. 90.—GRIT MILL.

closed at both ends, and lined with easily renewable chilled cast-iron plates. Two manholes *b* give access for introducing pebbles and replacing liners. Cast-steel naves riveted to the ends are provided with hollow journals, resting in strong bearings. The usual speed is 28 rev. per min. The ore is first reduced to 20 or 30 mesh in an ordinary ball mill, and is fed in through one of the hollow journals, being controlled by suitable mechanism *e*. The charge of pebbles for a drum $16\frac{1}{2} \times 4$ ft. is about 4 t. The discharge at *f*, like the feed, is by way of one of the hollow journals. The product is in this way kept remarkably uniform, and there is no risk of coarse matters or broken pebbles being mixed with it. The machine occupies a space of about 30 ft. long, 10 ft. wide,

and 10 ft. 6 in. high; weighs, without pebbles, some 12 t.; consumes 30 h.p.; and delivers per hour about 4 t. of a grade leaving only 5 % on a 76-mesh sieve.

*Stamps.*—In some instances, stamp batteries are in use for dry crushing. This would seem to be where the battery has been originally erected for wet crushing, and its conversion into a dry mill has been resorted to rather than the entire cost of the battery should be sacrificed, and an additional outlay incurred for dry-crushing machinery. This has been notably the case in New Zealand, and the great Waihi mill is a typical example. Here the stamps weigh 900 lb., drop 6 in., make 90 to 94 drops per min., and, in 1897, crushed 1·4 t. per head per 24 hours through a 40-mesh screen. According to J. McConnell,* the duty of stamps is much the same in both wet and dry crushing, and he gives the output at 1·75 t. through a 30-mesh screen crushing wet. But this comparison is entirely vitiated by the fact that a minimum of water (in reality cyanide solution) is used, which of course hinders the escape of the pulp. At the same time, he asserts that the wet-crushed pulp is of coarser texture than the dry-crushed product through the same screens. If that is so, then the duty per stamp of equal sized pulp must be greater with wet than with dry crushing; and in confirmation of this, it is said that in the early days at Waihi, when wet and dry crushing were being tried side by side, it was found that coarser screens could be used when dry crushing, viz. 30 to 40 mesh, as against 60 mesh in the wet battery. The explanation of this is quite simple—the duty in each case depends on the freedom of issue. The greater capacity of the wet battery necessitated the use of a much finer screen in order to retain the ore longer; in other words, the ore crushed in the dry battery did not issue freely (as it did in the wet) when it had reached a size represented by the 30 or 40 mesh screen, but was kept back and underwent additional comminution until it would pass a 60-mesh. The conditions are obviously forced in both instances. There cannot be any question that the addition of a powerful exhaust to the dry battery would materially increase its duty, and the only

* Trans. Inst. M. & M., vii. 26 (1898).

apparent reason why this was not provided is that fine crushing is the desideratum, and, if the issue were facilitated, then finer screens must be used. In the Waihi mill, the issue is dependent entirely on the disturbance of air and dust (finely crushed ore) caused by the fall of the stamps, and it is unnecessary here to insist that air in motion is much less effective than water in motion for causing an issue through the screens. The application of an exhaust to dry stamps would not be so successful as to a machine of more stable character; the vibration inseparable from a battery is bound to create endless leaks. But in any event, it would mitigate the intolerable discharge of dust throughout the mill. The author can say from personal experience that this is a most striking feature at Waihi. The site of the mill can be located from miles away, by the cloud of dust which envelops and hides it. Men who were spoken to assured him that many are incapacitated in less than two years, and a large proportion soon die from lung troubles. The waste of ore too must be enormous. But the slipshod style of the Waihi mill arrests attention at every step, and it gives one the impression that a ridiculously false economy in plant is the ruling spirit. Waihi certainly does not present an illustration of the best that can be done with dry-stamp milling, but rather shows up the defects of the system. It is said to have been adopted after trial of ball mills, in which case one can only suppose either that the ball mills were in bad hands; or that the fineness demanded by the Waihi ore could not be attained by anything but stamps, which, in the light of what is done elsewhere, cannot be accepted without question.

The Waihi Co.'s report for 1898 gives some interesting figures. The old 90-head mill, running 304 days 8 hours, crushed 47,018 t., and had a duty of 1·741 t. per head per 24 hours; the new (Victoria) 100-head mill, in 263 days, put through 40,261 t., giving a duty of 1·681 t. only. During about 2 months' work at the new mill, much trouble occurred from breakages of cam-shafts, and after trying different classes of steel, iron shafts are reported to have been found most suitable, and are to replace the steel ones as they come out.

Difficulty was also experienced in "getting cams and tappets to stand the very heavy wear due to quick running." The explanation may be traced to a preceding paragraph in the report, which says:—"Owing to the 'Sunday Labour Act,' which came into force on 1st January, 1898, our mills have not worked on Sundays. In order to make up, as far as possible, for this loss of milling time, the speed of the mills and height of drop have been increased to the highest possible limits. This has led to greater cost for repairs, but has considerably increased the stamp duty." One cannot but regard this as an excellent illustration of "how not to do it." The delusion of gaining a small extra duty per day of actual working, at the expense of prolonged and repeated stoppages to effect repairs and renewals, should be obvious to anyone, while the cost of these renewals must add to the item "maintenance" in the milling accounts. The idea of trying to crowd 7 days' work into 6 is opposed to ordinary common sense, and if it were possible, it would only prove that the mill, when working 7 days a week, was not allowed to fulfil its proper capacity. It is not often that such a good object lesson can be extracted from an official publication.

In the new mill, the ore is trammed to the kilns, passed through a gyratory breaker in a separate building, and trucked to the stamps, which are of 960 lb. and make 90 drops per min. The 100 stamps are arranged in a single row with the motors (turbines) in the middle. Using 40-mesh screens, the bulk of the product passes 80. There is both front and back discharge, into worm conveyors which deliver to the cyaniding house, 225 ft. distant. These conveyors are 15-in. steel tubes, mounted on friction rollers, and fitted with an internal fixed spiral. The tubes rotate by water power.

The Waitekauri mill has 40 stamps of 1000 lb., making a 6-in. drop 95 times per min., the duty through 40 mesh being about 1·792 t. per 24 hours. An elevator midway delivers the pulp to bins behind, from which it is hand-trucked to the cyaniding vats.

At the Crown mill, the dried (p. 283) ore is broken and

trucked to bins, when it is fed to the stamps and crushed to pass 30 mesh. Escaping through the screens it falls into a screw conveyor trough, which takes it to a bin, whence it is raised by a bucket-belt elevator and discharged upon an 8-in. rubber belt with rope edges, which traverses a hopper 110 ft. long running the whole length of the cyanide vat house, with a door and a track for hand trucks to each vat.

At the Waitekauri mill, N.Z., it is stated by A. H. Bromly * that the pulp from a ball mill with 60-mesh screens contained less slimes than that from dry stamps with 40-mesh; but here again is lacking the essential element of comparison, viz. evidence that the discharge from the battery was facilitated so as to reach its maximum. To handicap a machine, and then proclaim its faults, is not rational criticism.

J. McConnell † describes the usual practice in New Zealand to be as follows:—The ore, generally speaking, is exceptionally hard, and crushing by stamps is almost universal. After drying, it is put through a breaker, or preferably, both a breaker and a Gates crusher, so as to reduce it before feeding into the stamps. These latter weigh 900 to 1000 lb., and are driven at 94 blows a minute, with a 6½-in. drop. The screen mostly used is 30-mesh steel wire, and the duty is 1·12 to 1·40 t. per head per 24 hours. The grade obtained, when crushing through a 30-mesh screen, is—

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Passed 30 mesh | retained on | 40 | 1·20% | | Passed 60 mesh | retained on | 90 | 12·56% |
| ,, 40 | ,, | ,, 60 | 9·00 | | ,, 90 | ,, | . . . . | 77·24 |

Thus over 77 % of the ore product is crushed fine enough to pass through a 90-mesh screen. The duty of the Talisman 20-stamp mill in 1899 was 1·233 t.

*Sundry.*—Of late years, several new forms of dry-milling machinery have come into notice and received some attention.

The Griffin mill, Fig. 91, is installed at Delamar (Nevada), and at Kalgurli (W. Australia). In the former case, it is

* En. & Min. Jl., Nov. 12, 1898.

† Trans. Inst. M. & M., vii. 27 (1898).

said that the 30-in. size, weighing $5\frac{1}{2}$ t., and using 25 h.p., is crushing 1 to $1\frac{1}{4}$ t. per hour from 1-in. cubes down to 50 mesh at a cost of $7\frac{1}{2}$*d.* to 8*d.* per ton. These figures are quotations from the Cyanide Plant Supply Co.'s (London) catalogue. The machine consists of a single roller suspended from a vertical axis, and rolling on the inside of a die-ring at about 200 rev. per min. An air current draws the crushed material out through a screen and passes it to a screw conveyor and dust chamber. Excellent reports are given of the sharpness

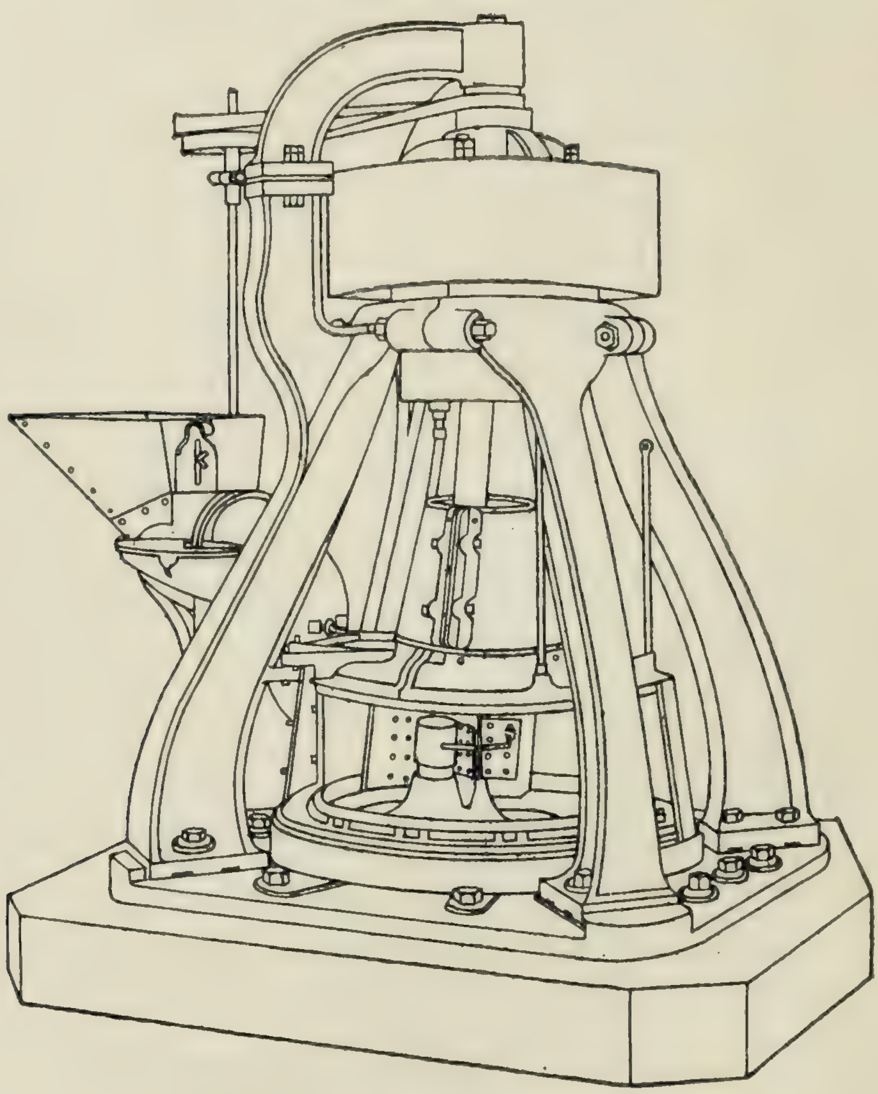


FIG. 91.—GRIFFIN MILL.

of the pulp produced and the absence of slimes, and quite a number of machines are now installed on the fields named. Their erection is obviously a simple matter in comparison with some machines. They are priced at about 450*l.* each.

The Niagara pulveriser has been tried * on Rand banket. A 7-ft. machine, receiving $8\frac{3}{4}$ t. per hour, and requiring (includ-

* F. White, Trans. Inst. M. & M., vii. 129, 134, 135 (1898).

ing screen) 21 i.h.p., accomplished the following duty in a 72-day run, passing over 20,000 t. of ore :—

| Free Milling Ore. | | | Pyritic Ore. | | |
|---|---|---|---|---|---|
| Size of Mesh. | Ore. | | Size of Mesh. | Ore. | |
| | Entering. | Leaving. | | Entering. | Leaving. |
| in. | % | % | in. | % | % |
| $+\frac{3}{4}$ | 51·2 | 0·00 | $+\frac{3}{4}$ | 93·5 | 0·00 |
| $+\frac{1}{4}$ | 29·3 | 64·30 | $+\frac{1}{4}$ | 6·5 | 80·5 |
| $+\frac{1}{8}$ | 3·7 | 7·70 | $+\frac{1}{8}$ | .. | 6·7 |
| $+\frac{1}{16}$ | 6·7 | 11·80 | $+\frac{1}{16}$ | .. | 6·7 |
| $-\frac{1}{16}$ | 9·1 | 16·20 | $+\frac{1}{20}$ | .. | 1·3 |
| | | | $-\frac{1}{20}$ | .. | 4·8 |

The wear and tear of path and roll amounted to 7554 lb. (of which 4808 lb. was path of defective shape that had to be discarded, though not really worn out), and the cost of wear was 2·258*d.* per ton milled. It will be understood that the mill was set to deliver a $\frac{1}{2}$-in. product, and this it did successfully.

**Cost of Dry Milling.**—In a presidential address, J. H. Collins* gives the average cost of roll-crushing hard ores to $\frac{1}{2}$-in. size at 6*d.* to 1*s.* per long ton ; Krom-roll crushing to 30 mesh, at 1*s.* 6*d.* to 3*s.* 6*d.* ; grinding hard rock in iron mills from $\frac{1}{16}$ in. down to 50 mesh, at 1*s.* to 1*s.* 6*d.* ; and the same to 80 mesh, at 1*s.* 9*d.* to 3*s.* 6*d.*

At the Waihi mill, in 1897, 87,670 t. were broken and stamped for 3*s.* 11·3*d.* per ton ; and in 1898, 86,696 t. for 3*s.* 11·5*d.* per ton. In the latter year, breaking cost 10·86*d.*, and stamping, 3*s.* 0·65*d.*

At Luipaard's Vlei, Transvaal, in 1898, the cost per ton for milling pyritic ore to pass 300 mesh was :—Breaking, 9·474*d.* ; white labour, 10·995*d.* ; native labour, 3·924*d.* ; stores, 8·863*d.*, lighting, 0·485*d.* ; steam power, 7·856*d.* ; electric power, 0·408*d.* ; maintenance, etc., 7·322*d.* Total, 4*s.* 1·600*d.* (Franklin White). The Lisbon-Berlyn is said to have milled

* Trans. Inst. M. & M., iii. 319 (1895).

28,555 t. at a cost of 1*s.* 3½*d.* per ton, but the mesh is not given.

As to comparative cost of wet and dry crushing, White * found the cost of wear in stems, heads, shoes, and dies in wet stamping to be 3·1*d.* per ton crushed, and the amount of wear (stems not reckoned) to be ·72 lb. per ton ; as against this, using two rolls and a Niagara pulveriser, the cost was 5·4*d.* and the wear was ·80 lb. ; while, "owing to the makers of the Niagara mill not having hit upon the proper hardness of the material," he had to "throw away about two-thirds of the weight of crushing surfaces before they were worn out." The claim is also made by White that he was able to dry crush with "fully 20 % less horse-power, and the saving in the fuel bill would more than compensate for the little extra wear and tear." The reduction in water consumption has already been mentioned (p. 274). The capacity of the plant is, however, not made at all clear. He says he was "able to crush through a 300 mesh at the rate of 9 t. per hour ; the plant was really equal to a 40-stamp mill of heavy pattern" ; and that the "total horse-power used was 86, which is much less than would be required for a 40-stamp mill with its pumps and tailings-elevator." In the next line, he says, "200 mesh gave about 85 % extraction," so that one is in doubt whether 200 or 300 was the mesh used ; but taking the better view of the case, viz. 300 mesh, or about 17 holes per linear inch, a well equipped modern battery should exceed the output ; and certainly would not exceed, even if it equalled, the consumption of power. So that the figures given by this pronounced advocate of dry crushing do not bear out the claim for its economy over the wet stamp mill.

**Examples of Practice.** *New Zealand.*—In the Waihi ore the gold is in an extremely fine state of division ; free gold is never visible. The richest stone has a chalcedonic look, the dark streaks carrying much sulphide of silver, and in them the most gold is found. The proportion of silver to gold is very large, the fineness of the bullion being about 640.

* Trans. Inst. M. & M., vii. 61–2 (1898).

F. Merricks * has described the changes in the milling methods here as follows :—

The first 2 t. were wet crushed and amalgamated on copper plates, the bullion saved being 35 % of the original value of the ore (75*s.*). During 7 to 8 years after this, 20,000 t. of ore were similarly treated for an average return of 12*s.* per ton. In 1890–94, a series of experiments were made on pan amalgamation, and both dry and wet crushing were resorted to, with the result that in the former case 70 % of the gold and 45 % of the silver, while in the latter, 60 % of the gold and 25 % of the silver, were saved. It is reported that the duty of the stamps in each case was practically the same, but by wet crushing the "calcined" (? dried) ore, the output was increased 30 %, while the extraction still remained 10 % in favour of dry crushing. In 1894, pan amalgamation gave way to cyanide treatment, the ore being dry crushed, with the result that an average extraction of 90 % of the gold and 50 % of the silver was effected.

The product of the Talisman mine is sometimes much mixed with clay, and contains, in addition to silver sulphide, nearly 1 % of iron and copper pyrites ; its bullion is only worth 17*s.* per oz.

At another New Zealand mine, the ore is very porous, and yields its gold quite readily when crushed dry to only ¼-in. mesh—a proposition quite unfit for a wet battery.

Some of the New Zealand mills, however, which have been for a long time wedded to dry crushing, are again turning their attention to wet milling. The Crown and Waitekauri have indeed adopted it, and the Waihi, the Woodstock and the Talisman have been experimenting. The latter, crushing through 30-mesh screens, with a ·85 % KCy solution in the mortars, obtained a 69 % extraction, and left only 7 % of slime, which assayed 7·29 dwt. gold per ton, as against 13·09 dwt. in the original ore. The percentage of extraction of the gold alone (which is the main object) reached 90, and Hooper's condemnation of the results seems to be somewhat hasty. At any rate, the official report for 1899, says "experi-

* Trans. Inst. M. & M., vii. 36–7 (1898).

ments have unquestionably proved that the ore is amenable to wet treatment. The substitution of wet for dry crushing will probably effect no saving in consumption of cyanide of potassium, and may not very considerably increase the recovery of bullion per ton of ore; but in view of the elimination of the cost of drying (fuel and labour), of the increased capacity of the stamps, of the material shortening which is likely to be effected in the time required for treatment of the ore, and of the reduction of cyanide royalty, owing to saving of bullion by amalgamation prior to treatment by cyanide, wet crushing should unquestionably be adopted in any new plant; and, when such new plant has been erected, the present mill should be converted to the same method."

The Waitekauri Co. reports that the whole of the 40-stamp mill has been converted from dry to wet crushing, and is working satisfactorily, the result being a better extraction of bullion at less cost, while it is expected that the mill will reduce a larger quantity of ore.

*Transvaal.*—Dry crushing (for direct cyaniding) at the Lisbon-Berlyn mill is thus described by M. T. Brown * :—"The first trial was made by screening off all the smalls, coming from the Blake breaker, which would pass a $\frac{3}{8}$- or $\frac{7}{16}$-in. bar screen. This amounted to about 30 % of the whole; it was taken direct to the cyanide vats, the balance continuing to be treated as before in the stamp battery. The success met with justified the extension of the direct-treatment method, and two Marsden fine crushers were introduced to deal with the coarse product from the rock breakers, so that all the ore might be dry crushed. When the jaw faces were new, these were able to reduce the ore to about the size of peas, which was fine enough; but the corrugations quickly wore off, and a much coarser product was the result after the first few days, which meant replacing the movable jaw faces every 3 or 4 days—a very heavy expense. This difficulty has been got over by adding a set of Gates rolls as a third stage in the crushing operations; these give a satisfactory product, and

* Trans. Inst. M. & M., vii. 51 (1898).

the shells need to be turned up only about every 6 weeks. The Marsden fine crushers, instead of being set to give a very fine product, are now an intermediate step in the process, and the jaw faces which had been discarded, only partly worn, can be put back into the machines and more duty got from them. The crushing is done in three stages, by Blake breaker, Blake-Marsden fine crushers, and Gates rolls. After each stage, the ore is screened or trommelled, so that what is already small enough does not pass on to the next stage, but is shunted off instead to the vats. When the direct treatment was first introduced, the ore was very porous, and a coarser product was permissible ; the necessity for three stages of crushing was not then so obvious. As it is drawn from further into the hill, it becomes damper, and clayey or earthy matter adheres to the surfaces of the crushed ore, and renders the particles less permeable to the liquor. Washing the ore before crushing has been suggested as a method of getting rid of the adhering envelope of earth or clay" ; but this would entail subsequent drying, and it would seem more rational to dry the ore in the first place.

In the opinion of Franklin White,* "crushing machinery should now be chiefly designed to produce a product suited for cyanide treatment ; there does not appear to be sufficient reason in many cases why amalgamation should be resorted to at all"—alluding to the Rand ores, of course. Acting on this view, he installed a dry-milling direct-cyaniding plant at the Luipaard's Vlei mine, and treated some 22,000 t. in that way ; but inasmuch as part of the work was of an experimental character, and the tests made were often carried to an extreme point (in order that very definite data might be obtained), the cost of the work and perfectness of extraction were affected. During the first 4 months, the ore treated was chiefly oxidised : the first month's residues ran 1·6 dwt. per ton ; they then decreased to 1·1 dwt. The last 4 months' work was on pyritic ore, and the residue assay rose to 1·77 dwt., afterwards dropping to 1·15 dwt. "The crushing

* Trans. Inst. M. & M., vii. 124 (1898).

capacity of the plant was so much in excess of what was expected, that it only ran at intervals. Such work could not be done economically."

The ore operated on was the ordinary Rand "banket" or auriferous conglomerate—a sedimentary deposit, consisting chiefly of more or less rounded fragments or pebbles of quartz, practically non-auriferous, and often considerably fractured by earth movements and pressure, the interstices between the pebbles being filled with finer sediment or "cement." Iron pyrites is also present in quantities varying from, say, 2 to 10 %. The gold is generally in the cement, on the outside of the pebbles, and in the iron pyrites. Silica has been subsequently deposited in the conglomerate beds, the originally porous layer being converted into a compact mass. Stringers and pockets of quartz are sometimes found near faults and dykes, containing sulphides of iron, lead, copper, and zinc; but they are generally very poor in gold. In a few cases, however, pockets of very coarse gold have been found. There is no gold in the quartz pebbles, which constitute at least 75 % of the whole mass, and there is therefore no need to carry the crushing beyond the point which frees the gold in the cement.

The crushing plant consists of two Gates breakers (Nos. 2 and 4), a Niagara mill (7 ft.), and two Gates rolls (Nos. 1 and 2). The ore from the grizzlies and sorting floors is fed to the No. 4 breaker, and runs from it to the No. 2, which can thus deal with 20 t. per hour; thence it goes through a revolving screen, and is separated into three sizes, $-1\frac{1}{2}$ in. $+\frac{1}{2}$ in., $-\frac{1}{2}$ in. $+\frac{1}{4}$ in., and $-\frac{1}{4}$ in. Each size is collected in a separate bin, the proportions, in the case of pyritic ore, being about—

| | Weight. | Moisture. |
|---|---|---|
| Pieces $-1\frac{1}{2}$ in. $+\frac{1}{2}$ in. . . . . . | 53·4% | 0·83% |
| ,, $-\frac{1}{2}$ ,, $+\frac{1}{4}$ ,, . . . . . | 23·8 | 2·50 |
| ,, $-\frac{1}{4}$ ,, . . . . . . . | 22·8 | 5·00 |
| | 100·0 | Av. 2·225 |

The ore is then raised in trucks to its proper mill bin, the top of which is 25 ft. above the mill platform, itself 10 ft.

above the ground. The Niagara mill and the Gates rolls rest on strong timber framing in the order named, receiving the $+\frac{1}{2}$ in., $+\frac{1}{4}$ in., and $-\frac{1}{4}$ in. ore respectively. The countershaft for driving them is fixed overhead on the bin floor; and a crawl, over each, facilitates removal of wearing parts. The Niagara mill is not fitted with its cone and pan for fine crushing, as it only reduces to pass $\frac{1}{2}$ in.

After passing under the roller, the ore is delivered into a large revolving trommel, which returns the coarse pieces, allowing the $-\frac{1}{2}$ in. to fall through a shoot on to a horizontal belt conveyor, 18 in. wide, running about 350 ft. per minute. The crushed ore from the two pairs of Gates rolls falls on to the same conveyor, and is carried to the elevator. This machine, 65 ft. between centres, runs at 300 ft. per minute, and consists of an 11-in. belt, with steel buckets every foot apart. Each bucket holds 9·12 lb. of ore when full. The ore is then discharged into shoots leading to two double octagonal revolving screens (fitted with bumpers), 10 ft. long, and tapered, the inner screen being 2 × 4 ft., the outer 3 × 5 ft., and 8 ft. long. The centres of the shafts are $19\frac{1}{2}$ ft. above the mill bin floor.

For pyritic ore, the screens are covered with woven wire (unvarnished) screening, $\frac{1}{4}$-in. mesh inside, and $\frac{1}{10}$-in. outside. Three products are thus formed: the $+\frac{1}{4}$ in. passing by a shoot to No. 1 roll bin, pieces of iron or steel being picked out on the way by an electro-magnet; the $+\frac{1}{10}$ in. goes to No. 2 roll bin; and the $-\frac{1}{10}$ in. falls into a trough with a short worm conveyor which carries it back to shoots leading to two similar screens below. These are covered with 200 mesh (say $\frac{1}{14}$-in.) sieving, and give two products, that which passes through being finished. The coarser portion goes to No. 2 roll bin, while the fine portion is taken by a worm conveyor to the "finished ore" storage bin, whence it travels by a 16-in. belt conveyor to the first row of tanks.

The 18-in. conveyor, elevator, and screens are all enclosed, hence the escape of dust was so little that the exhaust fans and settling chambers were not used. It was found necessary to take frequent samples of the finished product in order to

detect escape of too large particles through holes, or leakage from shoots.

Working on free-milling (oxidised) ore to $\frac{1}{8}$-in. mesh, 20 t. per hour could be finished. Pyritic ore could be delivered through 300 mesh at the rate of $8\frac{1}{4}$ t. per hour. With reasonable care, the ore can be mined dry enough to be crushed and screened without artificial drying. It is fed into the Niagara pulveriser at an average of $8\frac{3}{4}$ t. per hour.

The wear of rolls and pulveriser is in excess of what is considered good in stamps; but the pulveriser suffered from "defective shape and material of the path, which made it necessary to throw away a large quantity of steel still available for crushing. The roll shells wore down very well, and showed no tendency to become seriously grooved or hollow in the centres; it was not necessary to face them up at all, and No. 2, when finishing pyritic ore to 300 mesh, showed a perfectly fair surface." The combined weight of material worn out and discarded as useless was 18,160 lb., and its cost, per ton milled, was 5·794*d.*, of which the rolls were responsible for 3·536*d.* The belt conveyors and bucket elevator showed little signs of wear. The worm conveyor blades were originally made of too thin material, and did not last; this was easily remedied. No calculation "can be fairly made as to the cost of screens per ton crushed, but in normal work it should be less than in wet milling."

The power consumed in the mill totalled 123 i.h.p., viz. breakers, 26; lift, 11; Niagara mill and screen, 21; coarse rolls, 20; finishing rolls, 23; elevator, 6; screens, 4; screw conveyors, 5; belt conveyors, 6; electro-magnet, 1.

Screening of oxidised ore crushed to $\frac{1}{8}$ and $\frac{1}{10}$ in. gave no trouble when dry (1 % moisture), and but little up to 2·7 %; the latter, however, hung in the inclined shoots.

*West Australia.*—In the Kalgurli ores below 100 to 200 ft., the gold is much associated with tellurides and sulphides, sparsely disseminated through a mass of talcose and chloritic rock, by no means well adapted for wet crushing. Even the oxidised ores of the shallow zone do not lend themselves well to wet treatment, as wet stamping and plate amalgamation

return only 50 to 60 % of the gold contents ; and the resultant rich tailings contain a very large proportion of slimes, which are not amenable to the ordinary type of cyanide treatment, and involve costly special plant. Nevertheless, the latest idea is to revert to wet milling, and even to carry it to the extreme point of making all slimes.

# CHAPTER VII.

## CONCENTRATION.

THE term "concentration" is applied to various methods employed for separating the valuable from the worthless portions of the heterogeneous mass constituting the "pulp" derived from either wet or dry milling. The process of segregating the rich particles of "mineral" or "sulphurets" is dependent for its success upon their relatively high specific gravity. This is manifested in the accompanying list, giving the principal gold-bearing ores and the common gangues in which those ores are found :—

GOLD-BEARING ORES.

| | Sp. Gr. |
|---|---|
| Gold, native, pure . . . . . . . . . . . . . | 19·3 |
| Do., combined with antimony, arsenic, bismuth, copper, rhodium, and silver, in varying proportions . . . . | 10·0 to 16·0 |
| Tellurides of antimony, gold, iron, lead, mercury, and silver, containing gold up to over 44% of the bulk . . | 6·10 to 9·04 |
| Sulphides of antimony, bismuth, copper, iron, lead, mercury, molybdenum, silver, and zinc . . . . . . | 3·9 to 8·2 |

GANGUES OF AURIFEROUS ORES.

| | |
|---|---|
| Apatite, aragonite, calcite, chlorite, clays, dolomite, felspar, fluorite, gypsum, magnesite, muscovite, quartz, serpentine, and steatite . . . . . . . . . . | 2·3 to 3·25 |

The contrast is well-marked, especially as the heaviest of the gangues (fluorite, 3·25) and the lightest of the ores (zinc-blende, 3·9) are each of comparatively rare occurrence, and almost unknown as associates in the same milling dirt. It would of course, be easy to quote the exact sp. gr. of each mineral as given in the text-books ; but it must be remembered that such figures refer to the pure article in every case, and, while

they are interesting and useful in mineralogical study, the fact that ores are always more or less complex mixtures of many minerals makes a generalisation of the various classes of more utility to a millman than a detailed specification of each individual would be.

Up to a certain point, concentration resembles amalgamation, the object sought in each case being isolation and retention of the valuable particles while releasing the residue. They are alike, too, in this, that gravity is the operative force relied on. The difference occurs in the nature of the surface presented to the deposited particles, by which their arrest is completed.

Concentration as carried out on gold-mill pulp is a very simple matter in comparison with the niceties involved when dealing with base-metal ores. No differentiation of the products is necessary, all the valuable matters being collected together as "concentrates," while all the remainder escapes as "tailings." Almost anything which tends to diminish the velocity of the stream beyond the retardation due to friction between the nethermost stratum of the current and the plane over which it is passing, serves the purpose more or less well. Among the earliest appliances adopted were the undressed skins of animals, laid hairy side uppermost, and their use survives to this day. Irregularity in size makes them inconvenient of arrangement, and inequality of length and thickness of hair impairs the constancy of their work, so that blanketing or baize, made in pieces of suitable dimensions, has commonly replaced them where procurable.

It did not take long, however, for scientific observers to notice that something more than specific gravity has to be taken into account. The size of the particles has also an important influence, mass exerting itself in forcing a passage through the resisting medium (water or air, according as wet or dry concentration is followed) where gravity alone does not suffice. This has led to the provision of "sizing" apparatus as a preliminary to fine concentration.

Of considerable utility in this connection is the relative hardness of the gangue as compared with the ore. Thus

while the hardness of quartz, by far the most common gangue, is 7, and that of felspar is 6, the tellurides range from 1 to 3, and the sulphides mostly from 1½ to 4; and though ordinary iron pyrites varies between 6 and 6½, and mispickel or arsenical iron pyrites fluctuates between 5½ and 6, both are very brittle, more particularly the latter, which is as a rule the most auriferous of all sulphides. Thus the effect of the milling is to deliver the gangue ordinarily in much larger particles than the mineral, especially as the superior gravity of the latter hinders its freedom of issue from the mill, and subjects it to the risk of extra pulverisation, after the gangue has been reduced sufficiently to liberate it.

The proportion of mineral or sulphurets present in almost all ores is vastly greater than the percentage of free gold, so that, except where labour is abnormally cheap, economy of working demands that the operation of unloading the catchment area of the concentrator, and maintaining its potential activity, shall be as automatic as possible. Hence the introduction and success of various forms of concentrating machine.

Before proceeding to discuss these various appliances and their application, it will be interesting to observe, from the table below, the percentage of concentrates saved at a number of mills, and the wide variation in the value of their contents.

CONCENTRATES: PER CENT. AND VALUE.

| Mill. | Concentrates. | Concentrates: Gold contents per ton. |
|---|---|---|
| Brazil: | % | oz. |
| Ouro Preto . . . . . . . . . . | 2·0 | 2·5 |
| Do. . . . . . . . . . . | 0·07 | 30·0 |
| California: | | |
| Clinton . . . . . . . . . . . | 2·0 | 5·0 |
| Empire . . . . . . . . . . . | 2·25 | 4·0 |
| Gover . . . . . . . . . . . | 1·0 | 5·5 |
| Idaho . . . . . . . . . . . | 1·0 | 4·25 |
| Kennedy . . . . . . . . . . | 2·0 | 6·25 |

CONCENTRATES: PER CENT. AND VALUE—*continued.*

| Mill. | Concentrates. | Concentrates: Gold contents per ton. |
|---|---|---|
| California—*continued*: | | |
| Keystone | 1·5 | 5·5 |
| North Star | 3·0 | 3·0 |
| S. Spring Hill | 1·5 | 2·0 |
| Wildman | 1·5 | 3·0 |
| W.Y.O.D. | 2·5 | 5·0 |
| Colombia: | | |
| Remedios | 1·25 | 20·0 |
| Colorado: | | |
| Gregory Bobtail | 14·0 | 0·5 to 1·25 |
| Hidden Treasure | 13·0 | 0·75 |
| New York | 15·0 | 0·3 to 0·5 |
| Prize | 12·0 | 0·5 to 0·75 |
| Randolph | 20·0 | 0·5 |
| Dakota: | | |
| Homestake | 3 to 6 | 0·25 to 0·4 |
| New South Wales: | | |
| Lucknow | 2·62 | 24·27 |
| Transvaal: | | |
| Ferreira | 2·46 | 5·08 |
| Jumpers | 2·5 | 5·0 |
| Victoria: | | |
| Britannia | 1·0 | 1·2 |
| Catherine | 0·5 | 2·85 |
| Dixon's | 3·0 | 2·68 |
| Harrietville | 1·0 | 4·46 |
| Hillsborough | 2·0 | 1·34 |
| New Chum | 1·5 | 2·27 |
| North Cornish | 2·5 | 5·04 |
| Pearl | 0·5 | 2·14 |
| Port Philip | 0·75 | 3·61 |
| S. Clunes | 0·62 | 2·9 |
| S. St. Mungo | 1·75 | 1·6 |
| Star of the East | 3·5 | 3·08 |

**Crude Methods.**—Omitting the use of hides, which belongs properly to alluvial mining, and is of no interest to the millman, the application of blanket- and canvas-strakes and wooden tables only merits description.

*Blankets.*—The most suitable kind of blanket depends somewhat on the character of the pyrites—whether coarse or fine grained—but usually a closely woven texture is preferred. It is spread on an inclined wooden table, supported, adjusted and fixed on the same principles as the copper plate table (p. 228), which it sometimes replaces. The pitch is, however, reduced, ranging generally from $\frac{3}{4}$ in. to $1\frac{1}{2}$ in. per ft., and averaging about $1\frac{1}{16}$ in. At the same time, the pulp is made more fluid by addition of clean water. A distributing trough is often used to ensure an equal apportionment of the pulp over the whole table, which is absolutely essential to good work. For convenience in handling, the strips are made narrow, one or two stamps only being served by each; longitudinal fillets of wood about 2 in. sq. separate them, and provide foothold for the attendants. Ordinary lengths are 15 to 30 ft., requiring 3 to 6 strips, which are laid down (being first well soaked) as tight and smooth as possible, one overlapping another a few inches. Each division of the table is called a "strake." Sometimes the table is stepped into sections of 3 ft. or so, with 2-in. drops between. The table is best made so that its pitch can be altered while running. It is a common fault to make it too steep, in the anxiety to ensure clearance of the tailings. Water supply must be abundant and under entire control, its adaptation to the material under treatment being of vital importance. The interstices of the blanketing would in time become quite filled with concentrates, and would then cease to be effective; the unloading and washing of the blankets must therefore be done with sufficient frequency, according to the nature of the ore, predominance of fine pyritic matter demanding constant changing. Deposition takes place most rapidly near the head, and the first row of blankets may need washing hourly, or even oftener; the second row can wait for 2 hours, and the remainder for 6 to 12 hours. Carelessness is responsible for much loss, and the custom of

employing boys for this work because their wages are lower is very doubtful sort of economy.

It is not surprising to find blankets much in favour in the gold mills of S. American countries, where the wages of labourers are exceedingly low and where skilled workmen are difficult to find.

Thus in Brazil, at the Faria mill, blanket strakes have been added as supplemental to the vanners, the probability being that competent vannermen cannot be got. At Morro Velho, the washing of the blankets has been made to some extent automatic.

S. J. McCormick has described the elaborate blanket concentration methods at Ouro Preto very fully.* Here the blankets are attached to triangular revolving tables, and, following immediately after the battery screens, they are relied on to catch the free gold, which is washed from them and treated by pan amalgamation. Only the first 3 ft. of the revolving table carries blanket: after that come riffles for arresting the auriferous pyrites; these "revolvers" are turned every 20 minutes, and the concentrates are hosed into a launder which carries them to a tank. This mineral is then still further concentrated by means of a *passador* or wooden distributor, which feeds it to blanket strakes. The concentrates obtained finally carry 23 to 28 dwt. gold per ton. The pulp leaving the blankets passes over Frue vanners, where additional pyritic concentrate is saved, while a small amount of very fine free gold still escapes them, and is secured by blankets immediately following, the final tailings assaying about 1¼ dwt.

Similarly, in the Remedios district of Colombia, F. Owen † says:—After leaving the Frue vanners, the pulp passes over blanket strakes 12 ft. long by 2 ft. wide, with 4 divisions, and an inclination of 1 in. per ft.; thence into settling pits 5 ft. diam. by 1½ ft. deep. From ore containing 27¼ dwt. gold and 5 oz. 5½ dwt. silver per ton, the first blankets collected about 16½ dwt. gold and 2 oz. 16¼ dwt. silver, or 60 % of each metal,

* Trans. Inst. M. & M., v. 116 (1897)

† *Ibid.*, iv. 9 (1896).

sending on to the vanners a pulp containing 10¾ dwt. gold and 1 oz. 18 dwt. silver. The blanketing costs *8d.* a yard, and will last in use for 6 to 8 weeks. It is washed every hour, and sometimes every ½ hour. In the native mills, blankets extract 53 to 63 % of the gold, but, on replacing these mills by modern type installations, plate amalgamation has been substituted, the ore running 10 to 15 dwt. fine gold per ton.

Some of the Colorado mills place strips of blanket 3 ft. long by 18 in. wide between the amalgamating plates and the concentrating machines, to catch escaping mercury and amalgam, rusty gold, gold attached to particles of stone, and the heaviest pyrites (Rickard). They require washing every 2 to 8 hours, and it is difficult to appreciate their advantages over vanners.

New Zealand millmen are so wedded to blankets that a special article is made for their use, costing 12*s.* a yard (6 ft. wide). A single width covers a whole table, but it is divided into 4 partitions. The usual grade in Otago is ⅞ in. per ft., where green baize at 3*s.* a yard is preferred, and lasts 3 to 4 months. The newest blankets are put in the first row. Sometimes blankets precede the copper plates, measuring 12 ft. long by 4 ft. 3 in. wide (in 3 partitions), and at a pitch of 1½ in. per ft.

In Victoria, the blanket strake is almost universal, despite the high rates of wages, baize being often used. At Clunes, with a full width, and the unusual volume of water due to double-discharge batteries, the grade is only ¾ in. per foot. At Lansell's mills, Bendigo, the blankets are usually as wide as the box, and about 15 ft. long, sloping at 1¼ in. per foot; but concentrating machines are now being introduced with success.

A shaking blanket table has been designed by T. White,* which can be cleaned up with but little labour. A light frame suspended by 4 rods, carries 2 tables side by side, each about 9 ft. long and 22 in. wide. The slope can be regulated by means of turnbuckles on the supporting rods. The tables are covered with blankets, and there are 3 drops or steps in the

* Canad. Min. Rev., Nov. 30, 1898.

length of the table, which receives 200 longitudinal throws of $\frac{3}{8}$ in. per minute. When the blankets are charged with concentrates and gold, the feed is turned off by a trough to a spare table, and the tables are tilted sideways and washed off by a spray pipe. Such a double table weighs only 600 lb. If the frame has sufficient rigidity to maintain a true and even surface on the table, it may be an improvement on the ordinary form, as the ready method of washing must effect considerable economy of labour. But where tried, at Costerfield, Victoria, much trouble was experienced with the pulp packing in the corners of the table from the effect of the throw, so that its efficiency was much reduced.

*Canvas.*—For very close saving of the finest material, canvas tables have come into favour in some Californian mills, the article known as "10-oz. Arctic duck" being chosen. The principles and practice regarding it are similar to those of blankets.

At Ouro Preto, Brazil, McCormick had canvas strips 3 ft. long below the blanket tables. The pulp delivered to them assayed about 2·9 dwt. per ton, and the tailings leaving them carried still about 2·2 dwt., so that their efficiency is not very striking. The concentrates saved in them were worth about 2 oz. 17·37 dwt. per ton.

In many Hungarian gold mills a very rough canvas is used, and is regarded more favourably than either blankets or boards.

Among West Australian mills, the Ivanhoe is remarkable for having 50 ft. of canvas strakes immediately after the battery. These collect the sulphides, which amount on the average to about 2 %, and assay 12 oz. per ton.

*Boards.*—Wooden tables have their advocates. In Colombia, F. Owen passes the heavy tailings from vanners and blankets, after settlement in pits, over inclined (1 in. per foot) wooden tables, 12 ft. long by 4 ft. wide, having diagonal grooves cut by a rasp all over the surface. A small stream of water is allowed to play from the head of the table on to the tailings, which are continuously moved to and fro by a wooden scraper 10 in. long by 4 in. wide. The work is done by women and

boys at about 2*s*. a day, and is found to be very effective. A hard scrubbing brush, with an extra stream of clean water, is very handy for removing the accumulated material from the grooves.

Scratched boards have been used in Hungary for many years, but the concensus of local opinion regarding them is that canvas is superior.

**Sizing.**—Perhaps the simplest form of hydraulic sizer is the pointed box, the *spitzkasten* of the German millman, as shown in Fig. 92. These boxes are hollow rectangular pyra-

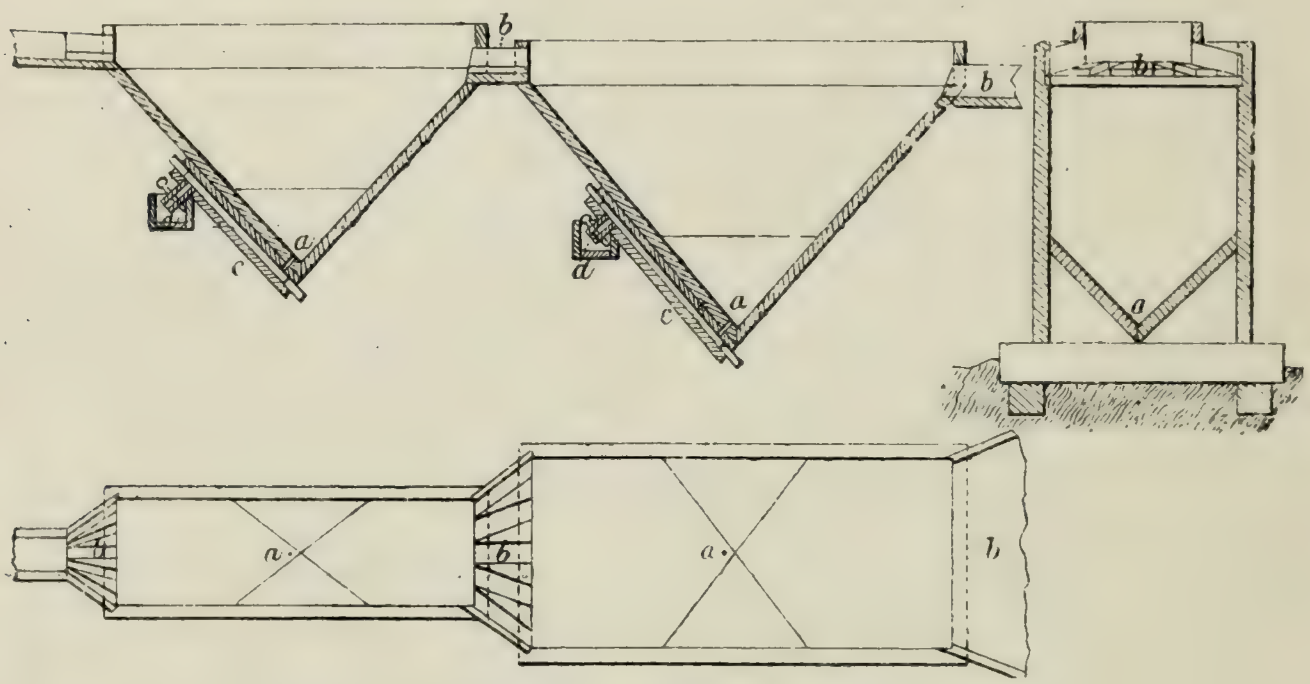



FIG. 92.—POINTED BOXES OR SPITZKASTEN.

mids, usually made of strong boards well jointed, or of sheet iron. The sides are inclined at an angle of not less than 50°, and a small hole is provided at *a* in one side, close to the apex. The pulp is admitted by a launder *b* at one of the narrow sides, a few inches below the top. As the box fills with pulp and water, a certain portion of the solid material, viz. the coarsest and heaviest,—which the water, on account of its diminished velocity, is unable to transport farther,—sinks and slides down the inclined sides of the pyramid, and escapes through the small aperture *a*, whilst the finer and lighter matters pass off at the top by an outlet and launder *b* in the centre of the side, opposite the point of entrance.

If now a second and larger box be attached to the first, a

third still larger to the second, and so on, each succeeding box at a slightly lower level to prevent any deposition taking place in the launders, it follows that not only the same process of settlement and escape of particles by the apex hole will take place in each box, but also that the size of the particles will decrease nearly in inverse proportion as the surface of the succeeding box is larger than that of the preceding one, or directly as the velocity of the water in it is diminished. According to this principle, if the boxes were made of only very gradually increasing dimensions, and the apex holes proportionately small, it would be possible to classify the pulp into a great number of portions, different in size of grain, before it had entirely settled, that is, till clear water passed away from the last box. Experience has shown, however, that classification into about 4 sizes is quite sufficient for practical purposes.

The dimensions of the different boxes, in order to ensure the most perfect classification, depend both on the amount of material which has to pass through them, and the size and character of the grains; and by theory and practice it has been found, that for each cub. ft. of pulp per second, the width of the first or smallest box must be $\frac{1}{10}$ ft., and for every succeeding box it ought to be about double that of the preceding one; or, in other words, the widths of the boxes must increase nearly in geometrical progression (2, 4, 8, etc.), and their lengths in an arithmetical one (3, 6, 9, etc.).

The depths of the boxes depend on the angle of inclination of the sides, which is generally about 50°; if less, the pulp is liable to settle firmly and choke the orifice; if more, unnecessarily great height of boxes is required.

The form of the two smaller boxes is commonly such that the short sides are inclined at the angle named, and the long ones, which would become far steeper, are broken, i.e. are vertical for a certain depth from the top, and afterwards inclined at the normal angle. This modification has, however, no influence upon the action of the boxes, but simply facilitates somewhat their construction and fixture in position. The sides of the larger boxes are generally even throughout.

The way in which the issue holes *a* are arranged has an important bearing on the operation of the boxes. At these points the hydrostatic pressure is considerable, and the holes should be kept small, in order to prevent too much water passing out with the pulp. Such small outlets are, however, especially in the treatment of coarser material, very liable to become choked. This difficulty has been met by the holes being made of conveniently large size, but connected with pipes *c*, ¾ in. diam., which rise up the sides of the boxes,—that of the smallest box to within 3 or 3½ ft., and those of the others to within 2 to 2½ ft. from the top,—and are there furnished with small mouthpieces *d*, supplied with taps for regulating the outflow. This arrangement, on account of the outlets being so much higher, possesses the further advantage that a considerable amount of fall is gained, especially as regards the large boxes, which, in view of subsequent treatment of the classified pulp, is sometimes materially useful.

Two other points demand strict attention, to ensure good action of the apparatus: the introduction of the pulp into the various boxes in a constant volume, and without splashing; and prevention of the entrance of wood chips, uncrushed ore, or any other substance likely to obstruct the outlets. The first point is met by having the supply launders expanded fan-like and furnished with dividing ledges, or by the interposition of small troughs, the sides of which nearest the box to be fed are perforated near the bottom by equidistant small holes. The cleaning of the pulp previous to its entering the first box is generally effected by the main supply launder being made a little wider near the point of entrance, and the insertion at this place of a fine wire screen, across the launder and somewhat inclined against the stream. This screen must be looked after, and freed from accumulated obstructions, but practically no other attention is demanded by the apparatus. Once in proper order, the action is constant and uniform, provided the pulp does not materially change in volume or character. The classified pulp can be directly conveyed without handling to any other desired treatment. A drawback, however, is the great fall of ground necessary between the

lower end of the copper-plate table and the feed boxes of the concentrating machines, in order to allow of the flow by gravitation from one to the other by way of this apparatus, so that frequently a sand pump or bucket-elevator is necessitated.

The action of the pointed box has recently been critically examined by R. H. Richards and C. E. Locke,* who find that it cannot be fed with such a product, or be run at such a speed, or be so adjusted to suit its feed, that it will do perfect work. There is always the tail-undercurrent to contaminate the product from the spigot. To cure this defect, three means have been considered, viz. a balanced hydraulic water-supply, an upward current of hydraulic water, and a perforated board. Much seems to depend upon the proper adjustment of current. Feeding with a horizontal current at the surface, above the surface, or with a plunging stream are all bad. To make the most of a settling-tank, the current of feed-slime must be brought to approximate rest as soon as possible after entering the tank, and a very slow movement must be established, which is of uniform velocity from top to bottom and from side to side. An even surface-distributor across the inlet-end, deep enough to slow the current greatly, and with a bottom steep enough to prevent the settling of slime, and this followed by two gratings made up of vertical bars 1 in. sq. with 1 in. spaces, the bars of the second grating staggered with those of the first, will probably prove satisfactory.

The original German invention has been much studied in America, and according to J. M. Adams, several forms are in use, their dimensions varying with the duty imposed. When, as is sometimes the case, it is desired to settle all the pulp, including the slimes, because too much water is present for effective concentration, the box is made 6 ft. deep and 3 ft. by 7 ft. at the top, the longest sides sloping till they meet at the bottom. Such a box will settle and save about 6 t. of ore in 24 hours, discharging it automatically and continuously from the bottom by a siphon hose, with the proper amount of water for subsequent concentration. This form is well adapted when pan-amalgamation tailings are to be concentrated.

* En. & Min. Jl., Feb. 20, 1897.

A pattern applicable to separating and saving slimes from battery pulp is shown in Fig. 93. Each box is 40 in. sq. at top and 40 in. deep, coming to a point at the bottom ; and one such box will handle 6 to 10 t. of pulp in 24 hours, making a good separation. The pulp enters at *a*, and is confined by partition *b* until it passes into the box proper *c* near its bottom. Clear water is admitted from a launder *d* above, through a ½-in. pipe *e*, which delivers it into the box at the bottom. Care must be taken that this pipe is kept full, so that no air is carried in, as the bubbles cause agitation, inducing a discharge

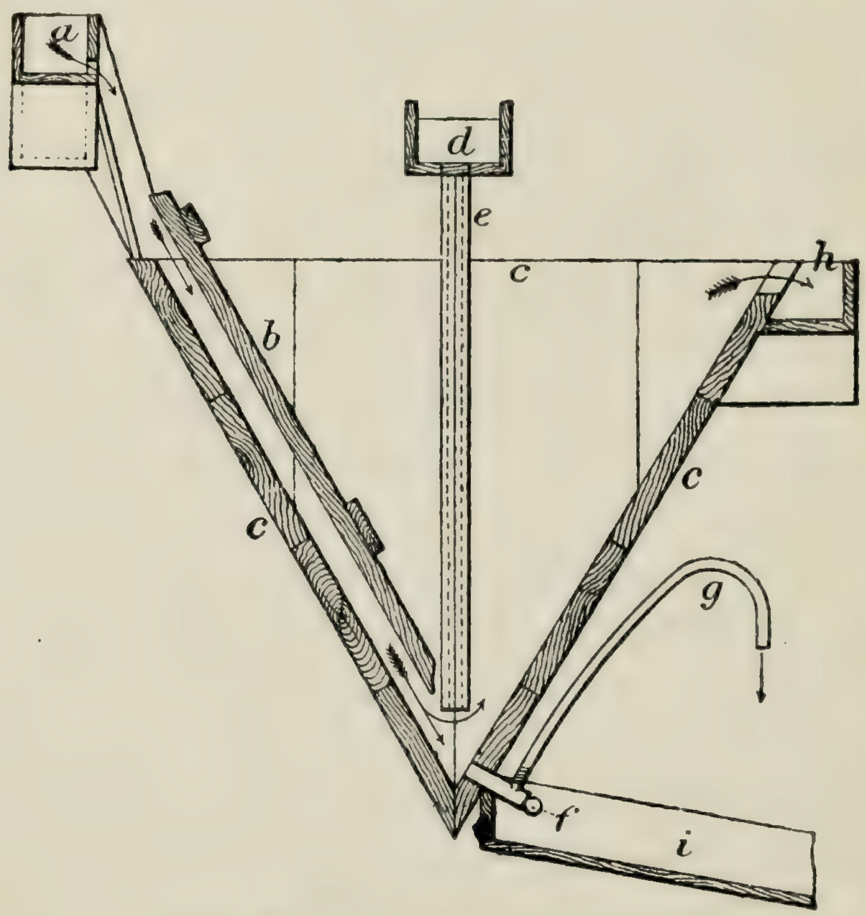



FIG. 93.—POINTED BOX.

of sand with the slimes. The amount of clear water consumed will vary, so the pipe should carry a cock just below the box *d*, or be controlled by plugging at its intake. To the hollow plug *f* is attached a piece of rubber hose *g*, which serves as a siphon, lessening the pressure and preventing too violent discharge. Without this siphon, a ⅛-in. opening would not be too small ; with it, ⅜-in. is about right. As, notwithstanding a screen over the box, coarse foreign matter does sometimes get in, the plug *f* may well be replaced by a 1½-in. iron T, with one end plugged, and with a ¾-in. side outlet, to which

the siphon hose can be attached. Launders *h* and *i* respectively carry off slimes and convey rich matters to the concentrators.

The pointed box is favourably regarded by some Transvaal millmen, and at the George Goch, for instance, one is placed at the foot of each copper plate table. The overflows are treated in 12 vanners, being one for each battery, while the concentrates issuing from the apex holes are distributed over 24 vanners. Thus is collected a quantity of more or less pyritic material which well repays cyanidation, while the final average tailings are reduced to an assay value of only 10 gr. per ton, which obviously leaves no margin for further successful treatment. Before the introduction of pointed boxes and vanners, the ultimate residues from the cyanide vats assayed $1\frac{1}{2}$ dwt.

In Fig. 94 is shown a new device made by Fraser & Chalmers, which is intended for use in connection with pointed settling boxes, to recover the pulp coming from Chilian mills, stamps, etc., in a thickened condition and of a consistence suitable for pan amalgamation, or subsequent systems of ore treatment requiring a continuous supply of thick pulp or tailings. A pointed settling box, 10 by 10 ft. by 8 ft. 6 in. deep, would be of suitable proportions for the application of the pulp extractor, securing even surface currents and a uniform slope of 60° for each side of the box. Such a box would have a delivering capacity of 8 to 10 t. of thick pulp per 24 hours. The construction and operation of the Ayton pulp extractor are briefly as follows :—A valve *a* is secured to the point of the settling box *b*, the movement of the valve being controlled by a lever *c*, held in position by means of a spring *d*, and actuated by a cam movement *e*. The extent of the valve opening and the frequency of the periods of discharge are regulated by adjusting mechanism. The pulp is allowed to settle or collect at the point of the box for a short period, when the valve opens and a portion of the thick settled pulp is allowed to escape, the valve being closed so quickly that the supernatant watery pulp does not reach the level of the discharge opening. This method of intermittent discharge

combines the advantages of a comparatively large opening with a limited flow of water, so that the thickness of the discharged pulp can be maintained, and the danger of obstruction of the pipes, together with the obvious disadvantages of a small opening under the full head of the settling box, which are attendant upon the use of a siphon discharge, are obviated. Results obtained at the Bote Reduction Works, Zacatecas, Mexico, show the efficiency of the device in actual practice.

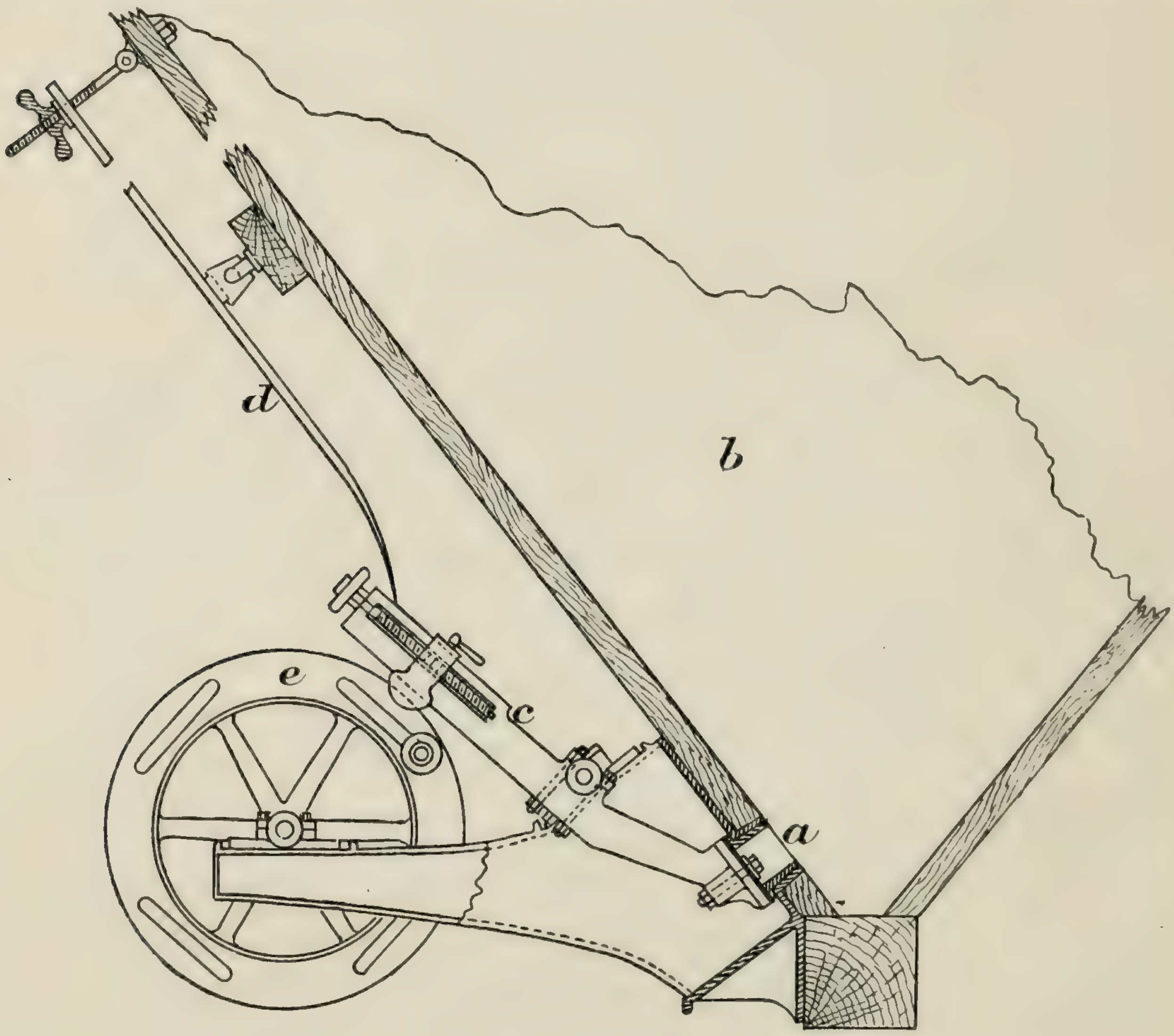



FIG. 94.—AYTON PULP EXTRACTOR.

The capacity of the mill is 70 to 75 t. per day; 6 Chilian mills effect the reduction of the ore, which requires admixture with 85 to 90% (by weight) of water. The thin pulp thus obtained is run into 8 settling boxes, which return to the Chilian mills 78 to 87% of practically clear water, while the pulp extractors deliver a continuous supply of pulp, containing more than 50% of dry ore, to the pans. A number of samples taken at intervals during June and July, 1899, showed a general average of 59½% dry pulp.

A step in advance of the pointed box is the triangular double trough or *Spitzlutte*, which is based upon the principle that if pulp containing particles of different size and density is exposed to a rising stream of water, the velocity of this stream may be so regulated that the larger and heavier grains will sink through it while the smaller and lighter portions are carried upwards by it; and that, consequently, by repeating this operation with gradually decreasing velocity of the rising stream each time, very thorough classification of the grains of pulp can be effected. The implement is shown in Fig. 95. Within a triangular trough *a*, with two opposite sides vertical and two inclined at an angle of 60°, is a similar smaller one *b*, having the vertical sides in common with the larger trough, but its inclined sides fixed at certain equal distances from and parallel to those of the latter. There is thus an open V-like channel *c* left between the inclined sides of the two troughs, representing, as it were, a rectangular pipe, sharply bent in the centre, and it is through this that the pulp for classification has to pass, that is, to fall and rise. The velocity of the stream depends on the size of this channel, as also does the size of the particles that will rise or sink in it. The cross section and respective velocity stand in inverse relation to each other; and their determination for each double trough of a complete apparatus is a matter of calculation, in which the size of the largest particles and the specific weight of the pulp to be classified form the main figures.

In order to carry off the coarse and heavy grains that sink in the channel, the inclined sides of the outside troughs do not meet below, but are continued downward, forming a long and narrow pyramidal opening *d*, about 1½ in. wide at top. The short sides *e* slope inwards at an angle of not less than 50°, contracting the aperture to a small hole *f* about 1 in. sq. at bottom, through which the material is discharged into a horizontal pipe *g*, that extends both ways a short distance beyond the sides of the apparatus, and is connected at the ends with 1-in. vertical pipes. One of these, *h*, serves for the outlet of the deposited grains, and is carried up to within 21 to 36 in. of the water level in the channel *c*, according to the fineness

of the particles that have to pass through it, as in the pointed box (p. 328). At the top it is furnished with a tap for regulating the outflow. The other pipe *k* conveys a supply of clean water from a launder *l* and cock *m*; and as the water in the pipe stands 6 to 8 in. above the water level in the trough, a slight but uniform pressure is produced, causing a forced influx of water at the point *f*, which is a most important agent in the classification process. This current, opposing itself to

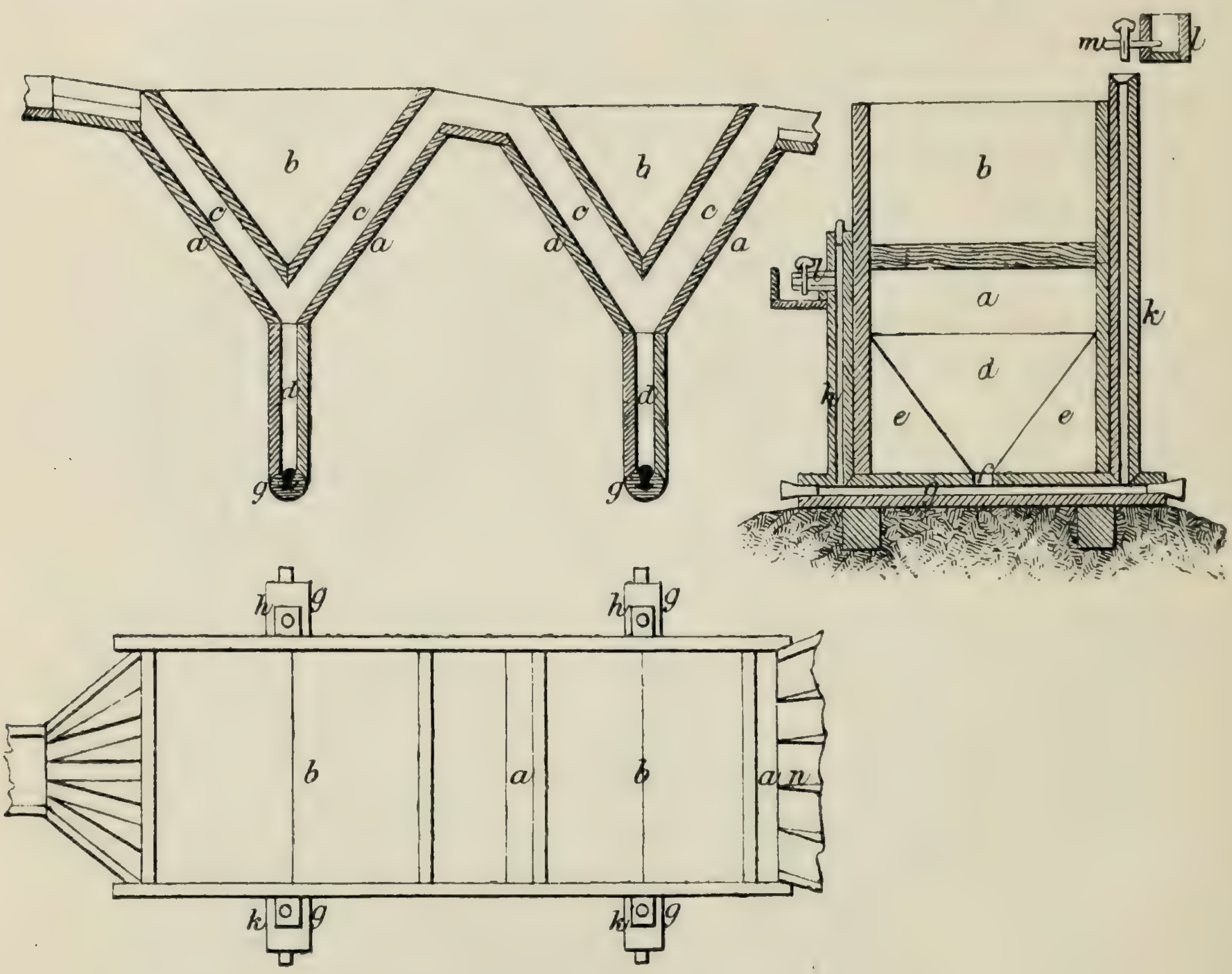



FIG. 95.—POINTED DOUBLE TROUGHS OR SPITZLUTTEN.

the downward stream of pulp and water in the channel *d*, prevents the issue of all but the coarse and heavy particles and clean water into the outflow pipe *h*.

The several double troughs of a series are fixed horizontally, and on a sufficiently descending level to prevent any settlement of pulp in the communication launders *n*, which are necessarily very broad. In other respects the working of double troughs much resembles that of pointed boxes (p. 324). In practice it is found that on coarse-grained pulp they give

better and cleaner products than pointed boxes ; but on fine slimes, the latter are generally preferred. A set of troughs requires less fall and space than one of boxes, and it is more readily adjustable to fluctuations of feed ; but the extra volume of clean water necessary constitutes an objection in some cases.

In the Transvaal mills, double troughs are coming into use of late. Thus at the Jumpers, in 1898–9, they were substituted for vanners, and 7288 t., forming 5·29 % of the tonnage milled, were caught as concentrates in these appliances, having an assay value of 15·34 dwt. fine gold per ton, as against 4122 t. of higher value by vanners. It is to be assumed that the extra cost of cyaniding nearly 70 % additional concentrates must be more than compensated for by the lessened cost of concentration by the simpler contrivance, and closer saving of values.

At the Crown Reef, 3 double troughs collect about 600 t. of concentrates per month out of 17,000 t. of ore, or about 3½ %. A 1½-in. pipe leads the heavy sands and pyrites, which assay 23 dwt. per ton, to settling tanks 30 ft. diam. and 6 ft. high. The main object of this rough concentration is to eliminate the coarse sands, so as to submit them to a prolonged treatment with cyanide. The cost of this system of concentration is only 25*l.* per month, mainly for pumping the water. The concentrates or classified material, being collected in wooden vats, is constantly under water, and does not get oxidised. The material collected is not very coarse, the battery-screens being 900 mesh.

Brown's "hydrometric sizer," a Californian modification of the German trough, much simplified and reduced in bulk, is very easy of control and most efficient in operation. It consists of a series of 3 or 4 galvanised iron basins, of conical shape with vertical rims, joined by communicating launders of similar material, the whole arranged as shown in Fig. 96, and supported by a light wooden framework. The basins increase in size as they become more distant from the original inlet of the pulp, though the perspective of the photograph would lead to the opposite conclusion. The first is just sufficiently below the discharge from the mercury trap at the foot of the copper

plate table to allow the pulp to gravitate into it. A short length of $1\frac{1}{2}$-in. pipe is attached to the apex of each basin, and

FIG. 96.—HYDROMETRIC SIZERS.

these are all united to a similar pipe conveying a stream of clean water. Close to the junction of each basin with the clean water main, is an outlet pipe for discharge of classified pulp,

which may be led from it to any desired concentrating machine. At each clean water inlet is a cock and pointed indicator for controlling the supply, and on that control depends the degree of sizing. Some experiments were made by the author at Lucknow with this apparatus, which proved it capable of dealing easily with the pulp from 10 stamps having a duty of about $1\frac{1}{2}$ t. per 24 hours, the first basin being only some 12 in. diam., and the third 18 in. Thus four classified products were obtained—everything larger than about 25 mesh from the apex of No. 1 ; 25 to 40 mesh from No. 2 ; 40 to 60 mesh from No. 3 ; and slimes, forming more than half the whole volume in this case, from the overflow of No. 3, which was split by the two-way launder shown, and distributed over two vanners. The sizing was very well done, without any attendance whatever, so long as the flow of battery pulp was even and constant, and a regular head was maintained in the tank supplying clean water. This latter in particular is a matter demanding the greatest attention if an undesirable excess of water in the slimes is to be avoided. It being, at Lucknow, impossible to devote more than 4 vanners to the 10 head, the benefit of the apparatus could not be availed of, because 2 vanners did not suffice to deal with the large volume of slimes, and on the whole poorer tailings were got by distributing the volume over 4 vanners, than by sending sands to 2 and slimes to the other 2. But enough was seen to prove to the author's satisfaction that, where facilities exist for apportioning the loads on the vanners, much better work can be done by them with preliminary classification than without, though a great deal depends on getting millmen who understand their work and take an interest in it.

Some experiments carried out by W. Bettel and J. H. Johns,* on behalf of the Johannesburg Chamber of Mines, point at first sight to an opposite conclusion. The idea was to roughly classify by means of spitzkasten and spitzlutten, and to pass only the pulp from the latter over the vanners. It was found that the vanners did less satisfactory work than

* En. & Min. Jl., Jan. 23, 1897.

with unclassified pulp, apparently owing to the following causes :—

(*a*) Overloading.

(*b*) The fact that minute particles of high specific gravity will subside through a shallower layer of pulp suspended by agitation in a turbid medium (such as a mixture of coarse and fine sand, pyrites and clay) more readily than through a classified pulp consisting of coarse particles suspended in water. This may be explained by assuming a reduction of surface tension in the former case.

(*c*) The proportion of fine gold escaping in sands and slimes is increased when an excess of water (over normal) is present in the pulp.

(*d*) With spitzlutten, an increased clear-water supply carries off fine gold and floured amalgam in considerable amounts in slime pulp.

(*e*) The scouring action of the coarse sands, etc., from the spitzlutten (free from slimes) prevents the collection of floured amalgam on the shaking amalgamated plates on the vanners, and this amalgam so lost is not wholly recovered in the concentrates collected.

The pulp (consisting of slime and finest sands) delivered by the spitzkasten is only imperfectly concentrated on the third vanner, such pulp requiring a large area equal to a very shallow layer of liquid for efficient concentration. The finest slimes (solid matter in effluent from spitzkasten) are very rich in gold ; this is attributed to the presence of suspended floured amalgam and free gold of extreme tenuity, probably coated with films of grease or sulphides, which cause the particles to exert an increased surface tension sufficient to overcome the action of gravity to a very considerable extent.

When Frue vanners are used for concentration, previous classification, in the opinion of the experimenters, would not be advantageous ; but where any other concentrator, not having a vanning motion, is employed, they would recommend a previous classification of the battery pulp.

In the author's opinion, the question is not settled by these experiments. It can only be determined where the vanner's

capacity is such that each machine shall receive a load that it can effectively deal with, and no more. Further, the adjustment of water supply to each vanner has to be nicely determined in order to get the best results.

**Concentrating Machines.**—While the number of concentrating machines introduced at various times is very great, comparatively few have ever attained any degree of success, or come into general use.

*Frue Vanner.*—Of all mechanical methods of separating a small bulk of fine-grained rich material from a large mass of refuse, or comparative refuse, the Frue vanner has long stood unrivalled for effectiveness, and it may properly occupy first place. Almost every year witnesses some minor alteration of its details, but the example shown in Fig. 97 indicates the general type. It is made by Fraser & Chalmers, Parke & Lacy, and the Gates Iron Works.

Main rollers *a*, carrying the endless rubber belt *e*, and forming the ends of the table, are 50 in. long and 13 in. diam.; made of No. 16 sheet iron, riveted lengthways, they are light yet strong, and are secured at the ends by rivets to light cast-iron frames. They are crowned about $\frac{1}{4}$ in. in the centre, and galvanised all over when finished, so that even the rivets are protected from rust. Complete, they weigh 70 lb. each. The bolts which fasten the boxes of *a* to the ends of *f* also fix on *f* the chilled cast-iron supports of the flat iron bars *n*. The rollers *b c* are of the same diameter, and are made in the same way as *a*. The rubber belt *e* passes through water underneath *b*, and is thereby made to deposit its collected burden of concentrates in the box 4; then emerging from the water, it goes over the tightening roller *c*. Rollers *b c* are hung to the shaking frame *f* by straps *p*, which swing on the holding bolts. Hand-wheels enable *b* and *c* to be adjusted, thus tightening and controlling the belt. The bearings of the rollers *a* have slots, so that, by drawing out or shortening, *a* can be used as a means of tightening the belt *e*; and as *e* sometimes travels towards one side, this tendency can be most easily checked by lengthening or shortening one end or other of *a*. The swinging

of *b* and *c* out of line also controls the belt somewhat. Supported by the small galvanised-iron rollers *d*, the belt *e* assumes the form of an evenly inclined plane table, the inclination being about $\frac{1}{4}$ in. to $\frac{1}{2}$ in. to the foot. This table, continuously moving forward, and at the same time receiving a cross vibration, has an ashwood frame *f*, bolted together, and with rollers *a* as its extremities. This frame is braced by 5 cross-pieces, shown

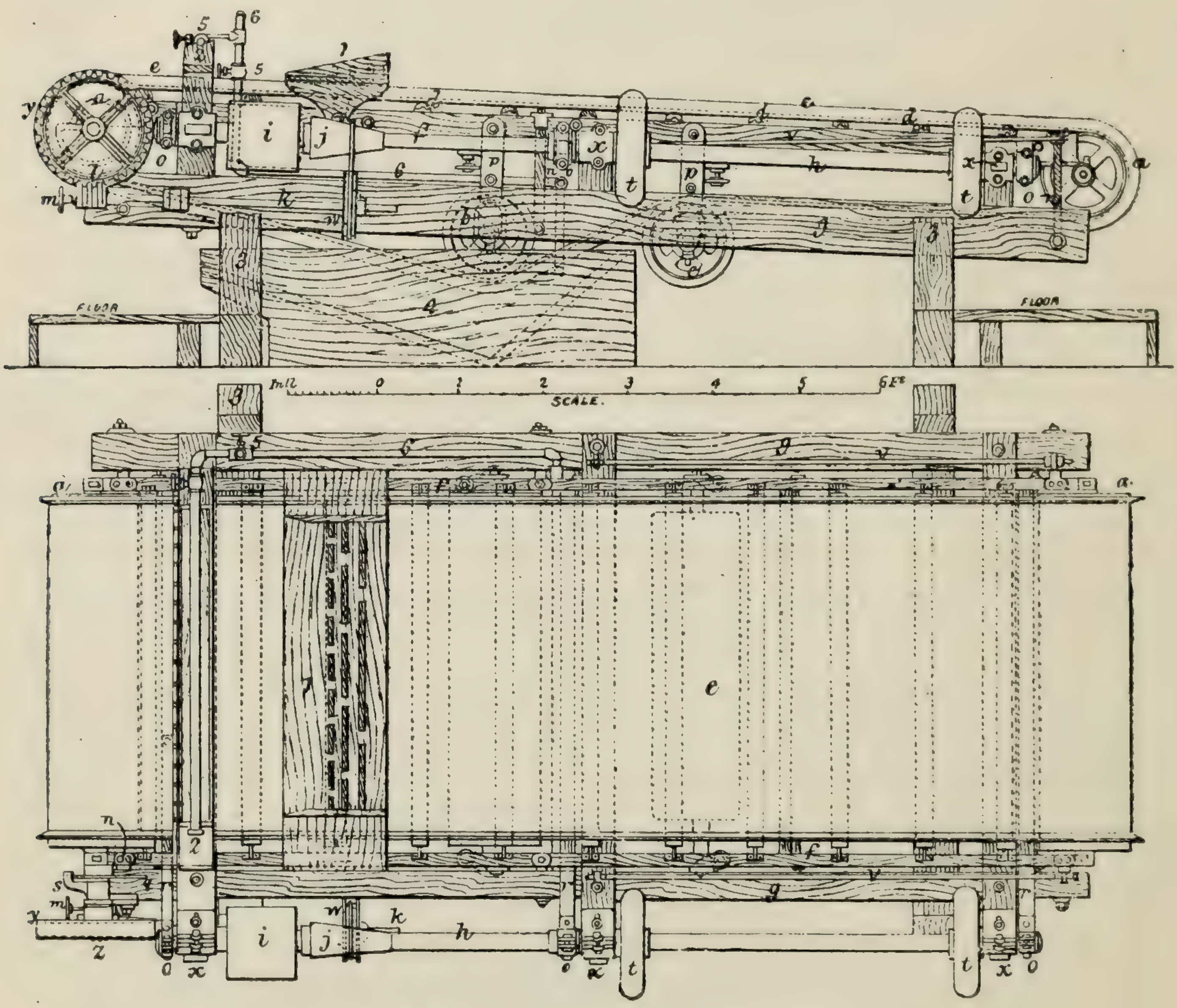


FIG. 97.—FRUE VANNER.

by dotted lines. The bolts holding together the frames pass through the sides close to the cross-pieces; the latter are parallel with *a* and *d*, and their position can be understood by the 3 flat spring connections *r o*, which are bolted to 3 of them underneath the frame. The belt *e* is usually 4 ft. wide, but a 6-ft. belt has latterly come into favour; its total length is $27\frac{1}{2}$ ft. It has raised edges, which will be further described.

Stationary framing *g* is bound together by 3 cross-timbers,

which are extended on one side to support the crank-shaft *h*. The inclination of the belt *e* is controlled by elevating or depressing the lower end of frame *g*, by means of wedges, as it rests on uprights 3, fastened to two sills which form the foundation of the whole row of machines in use. Frame *f* is carried on frame *g* by 3 uprights *n* on each side. These uprights are of flat wrought iron, drawn to a knife-edge at each end, and case-hardened, with chilled cast-iron bearings above and below. Each middle bearing on *f* (one on each side) has a single bolt hole, and the end ones (two on each side) have two. These bolts pass through frame *f*, and hold to the frame the bearings of *a*, which work in a slot. The bearings of the head roller are higher than those of the foot roller, that is, *a* is a trifle higher than the regular plane of the table ; and the first small roller *d* should be raised a little.

Bearings *x*, resting on the extension of the cross timbers of *g*, carry the main or crank-shaft *h*, the crank being ½ in. out of centre, and thus giving 1-in. throw. Power is applied through pulley *i*. The up-hill travel of the belt *e* is increased or diminished at will by shifting the small leather band connecting the cone-pulley *j* on crank-shaft *h* with the grooved pulley *w* on shaft *k*, a hand-wheel controlling the movement. The bearings of *k* are fastened to the swing box *y*, a cast-iron shell protecting the worm *z* and worm-gear *l* ; *y* turns on a bearing bolted to the outside of *g*, and thus becomes a swing box for swinging *w* and *k*. The object gained by this is that the weight of *w* and *k*, swinging with *y*, hangs on the small leather band and keeps it tight ; before this modification, the band was constantly slipping or breaking. A hand-wheel *m* is used to relieve the band from part of the weight of *k* and *w* ; by screwing it up, *k* and *w* can also be raised, taking all the strain off the band, and thus stopping the uphill travel. Shaft *k* terminates in the worm *z*, which connects with the worm-gear *l*.

Revolved by *l* is a short shaft terminating in an arm *s*, which drives a flat steel spring *q* (which is a section of a circle), connected with the gudgeon of *a*. Flat steel spring connections *r*, 3 in number, bolted beneath the cross-pieces of *f*, and

attached to the cranks of *h* by brass boxes *o*, give a quick lateral motion to the belt *e*—about 200 throws a minute ; fly-wheels *t* steady the motion. Rods *v* pass from the middle cross-timber to the upper bearings of the lower uprights *n*. The cast-iron washers on the bolts of the cross-timber have lugs cast on them, and so have the bearings of the lower *n*. Rods *v* pass through these, and have nuts on each side, thus preventing the whole movable frame *f* from sliding up or down, and enabling *f* to be squared.

Clean water is distributed by a wooden trough 2, which is supplied with water by a perforated pipe ; the water trickles on to the belt *e* by grooves $1\frac{3}{4}$ in. apart. The ore-spreader 1 moves with *f*, and delivers the pulp evenly to the belt. Various forms of distributor, however, are in use.

Upright posts 3 support the wooden frame *g*, but iron standards and frames are also employed. The water supply in 4 is regulated by cocks 5 from pipes 6.

In setting up the machine, everything must be in line except the tightener roller *c*. This can only gradually be adjusted as the belt *e* assumes its true position.

It will not be difficult to realise that the Frue vanner is a complicated and delicate machine, demanding constant care and attention, and capable of doing good work only under the most favourable conditions. It will not tolerate neglect or over-crowding, and the man who can properly manage more than a dozen of them must be smart and methodical, and moreover, have every convenience to his hand. The feed must be maintained with the utmost uniformity, both in volume and distribution on the belt, and the clear-water supply must be carefully adjusted to the needs of the case. A depth of $\frac{1}{2}$ in. of pulp on the belt can rarely be borne, and must never be exceeded. The motor actuating the machines must be sensitive of control, and have no other duty whatever that is liable to render its speed irregular. The uphill travel or advance of the belt will vary from 3 ft. to 12 ft. per minute, and the grade or inclination from 4 in. to 12 in. in 12 ft., according to the nature of the ore. Occasionally pulp will be delivered with too much water for satisfactory work, and in

that case a pointed box (p. 324) must be interposed to reduce it, with the likelihood that the effluent water will carry away some slimes that may possess high value and demand settlement for subsequent treatment. A certain portion of the area of the belt does practically no work, viz. a strip of varying width, generally about 6 in., along each margin. Here the ore manifests a tendency to pack, and separation of mineral from waste is very much impaired or entirely absent. It seems impossible to cure this evil altogether, and in order to minimise its ratio to the active part of the belt, the 6-ft. belt has been introduced as an improvement on the 4-ft.

The box 4 in which the arrested concentrates are washed from the travelling surface is kept full of water, but this does not always suffice to cleanse the belt completely, and a spray is often applied to the belt at its point of issue from the bath. This causes an overflow from 4, which must be conserved, as it will certainly carry fine mineral with it, and the finest mineral is mostly the richest. An adjunct for hastening the settlement of the concentrates in the box is a hammer automatically striking rapid blows on the side of the box. This is much favoured in the big Alaskan mills, and is called the "Alaska kicker." The concentrates box 4 is emptied at intervals, usually once in 24 hours, by scraping the contents out with a hoe into a receiving box placed alongside. This demands plenty of elbow room. A very convenient arrangement is to place the vanners in two rows, as shown in Fig. 96, the concentrate boxes being opposite each other, and a small watertight universal-tipping truck running on rails between them. In any case, however, the emptying of the concentrate boxes is a tedious and messy job, and forms a weak point in the machine. The effluent tailings flow away automatically into a general launder, which may be very simply made in the floor if, as should be, that is laid in concrete and faced with cement. A 9-in. × 2-in. plank forms a suitable cover for the launder, which must be given abundant grade, say $\frac{3}{4}$ in. per ft., to avoid choking.

The concentrating belt is of canvas stoutly coated with rubber. Generally the surface is quite plane and smooth, but

improvements have been sought in modifications of this. A so-called " doe-skin " belt, introduced by one maker, has an abundant nap, which certainly results in a marked increase of duty while the belt is comparatively new. This style was tried by the author at Lucknow, and for a few weeks the tailings from it were quite barren, while neighbouring belts of the ordinary kind lost 4 to 5 dwt. per ton. But once the nap wore off, the belt was much less efficient than the others. It would seem, however, that where classification can be carried out before vanning, this doe-skin belt (by Dick, Glasgow) may be advantageously used in dealing with slimes, as its capabilities would then be most useful, and its endurance prolonged. In first cost, it is slightly dearer than the normal rubber belt.

A corrugated belt has lately been introduced by Fraser & Chalmers, which is claimed to have an important effect, "allowing the use of steeper grade and more water, and, as a consequence, greatly increasing the capacity. Actual comparison during many months of steady working between machines with plain and those with the special belts in the same mill, have shown that one of the latter will treat as much pulp as two of the former. It is not necessary to point out that the saving in first cost, freight, erection, space, power and attention, by the use of one machine in place of two, is highly important. The improved belts are much more expensive than the smooth ones hitherto used, and have a much heavier rubber surface. They are found however, to be durable in use."

Durability of the surface of the belt is mostly controlled by the amount and quality of the rubber used in its manufacture. The cheapest article is usually very much the dearest in the end. The life may be as short as 3 months, and as long as 5 or 6 years, even on the same ore. A rubber paint is supplied by makers of belts, consisting of rubber dissolved in benzine, for application to the surface where it shows abrasion, and for mending accidental damages ; but in the author's experience this is not particularly successful, especially in the dry climate of the Australian interior. As a matter of fact,

however, the determining factor of the useful age of the belt seems to lie rather in the rim or flange than in the belt itself. Apart from careless running, in allowing the belt to overhang on one side, which may easily result in stripping the rim off forcibly, there are two sources of weakness. The first lies in the manner of attaching the rim to the belt. Owing to the unequal strains caused by the rim having to describe a larger circle than the body of the belt when passing round the rollers, the rim cannot be made an integral part of the belt, and the canvas foundation cannot be carried into it. Thus the rim is a separate structure, and is only "stuck on" as it were by rubber solution. Sometimes the adhesion is not too firm, and bodily detachment will occur long before the belt itself is at all worn. The second cause of premature incapacitation is the incessant alternation of expansion and contraction according as the belt is running flat or passing round one of the

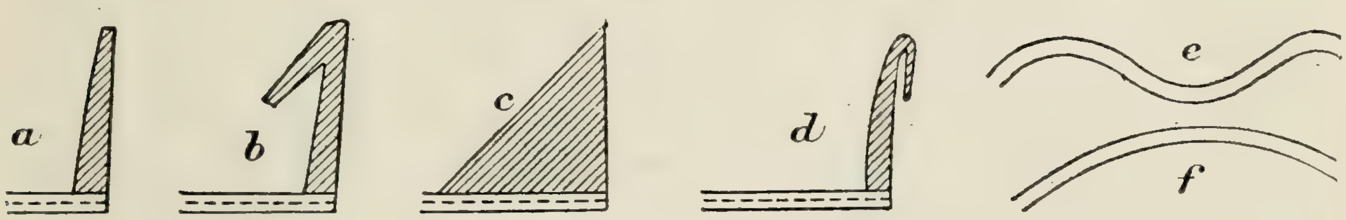



FIG. 98.—VANNER-BELT EDGES.

rollers, which has a tearing action on the extreme outer edge of the rim. With the object of reducing these evils, many styles of rim have been tried, some of which are shown in Fig. 98. Perhaps the most common pattern is *a*, in which the tendency is to fall outwards as the stiffness of the material lessens from repeated stretching, and also to split downwards and across; in *b*, splitting is not lessened, and soon the weight of the overhanging flange or barb causes the rim to fall inwards; *c* is the form adopted in Dick's doe-skin belts, and being of massive but high-class rubber it wears remarkably well; the "lip-flange" *d*, recently introduced, is next best to *c*, the folded edge manifesting much less tendency to crack than the sharp edges of *a* and *b*; the corrugated pattern *e* is a total failure, as in a very short time the corrugations merge into each other, as at *f*, and the mischief is intensified. With an imperfect rim, much slop and waste of pulp must occur,

as well as great detriment to the little rollers (*d*, Fig. 97) and their bearings.

Each machine, without belt, costs about 100*l*., and each belt about 30*l*., for the 4-ft. size.

The new 300-stamp mill of the Alaska Treadwell Co. has 120 6-ft. Frue vanners in a building by themselves ; they are ranged in 4 rows of 30 machines each, and are driven by 2 24-in. Pelton wheels.

In Brazil, at the Ouro Preto mill, Frue vanners catch about 30 t. a month of highly pyritic concentrate assaying 35 to 51½ dwt. per ton. According to McCormick,* "a small amount of very pure free gold still passes the Frue vanners, and is caught in blankets placed immediately after them—the final tailings assaying about 1¼ dwt. per ton." This seems to imply very bad handling of the Frue vanners. Again, at the Faria Co.'s mill,† "a very large area of canvas strakes was built, over which the tailings from the Frue vanners, diluted with clean water, were led ; a good concentrate, assaying about 25 to 30 dwt. per ton, was obtained on these strakes, but the quantity of sand caught was small, and the gold retained was a mere fraction of the total in the slimes." Therefore concentration in any form was abandoned ; but the asserted superiority of canvas strakes over Frue vanners demands investigation into the circumstances before it can be accepted as conclusive.

Californian millmen, who are among the most advanced, after trial of Frue, Triumph and Woodbury vanners, prefer the first as needing least attention. At the typical North Star mill, according to P. R. Robert,‡ the Frues have a slope of $\frac{3}{16}$ in. per ft., make 200 vibrations a minute, and travel 24 in. per minute. The film of pulp is $\frac{5}{8}$ in. thick. Repairs, exclusive of belts, in 5 years, averaged 28*s*. 9*d*. per machine. They have been in use 9 years, and during that time half the belts have been replaced. The attendants get 12*s*. 6*d*. a day, one man watching 16 machines and working a 12-hour shift.

* Trans. Inst. M. & M., v. 119 (1897).

† Official Report, 1898.

‡ Trans. Inst. M. & M., v. 155 (1897).

The cost of concentrating is $6\frac{3}{4}d.$ per ton. On a year's work (about 20,000 t., assaying 5·91 dwt.), the Frues collected 2·75 % concentrates assaying 38·16 dwt., and the final tailings assayed only ·68 dwt.; practically the whole of the concentrate is iron pyrites, mere traces of galena and blende being the sole additional sulphurets. The relative fineness of concentrates and gangue is shown in the following table:—

| No. of Screen. | Concentrates: % left on screen. | Gangue (tailings): % left on screen. |
|---|---|---|
| 30 | 4·10 | 13·77 |
| 50 | 4·80 | 12·00 |
| 60 | 10·83 | 22·55 |
| 80 | 11·11 | 11·71 |
| 100 | 5·77 | 7·76 |
| 120 | 6·27 | 4·65 |
| 150 | 13·08 | 10·00 |
| Finer than 150 | 41·16 | 15·85 |
| Loss, dust | 2·88 | 1·71 |

Thus, while over 48 % of the gangue would not pass a 60-mesh screen, 57 % of the sulphurets passed 120-mesh.

At the Lucknow battery, New South Wales, 2 Frues are provided for each 5 stamps, and they receive pulp assaying ½ oz. to 3 oz. gold bullion per ton and carrying 2½ to 3 % of concentrates. When, as sometimes happens, pyrrhotite or mundic prevails in the ore, the concentrates will range in value between 6 and 8 oz. per ton (bullion = 850 gold, 150 silver) only; but with the normal proportion of mispickel, 60 to 100 oz. is often reached, and, on a single occasion, the phenomenal assay of 172 oz. bullion per ton was registered for 24 hours. But an effort is always made to maintain a fairly uniform grade of 20 to 25 oz. per long ton. The ratio of tailings assay to concentrates assay, from samples taken automatically every 30 minutes, with monthly checking of daily records, is preserved with singular regularity by the vanners, notwithstanding continually repeated fluctuations in

the value of the pulp fed to them. Following are the respective figures for 5 years' work :—

| Concentrates, assay. | Tailings, assay. |
|---|---|
| 29·14 oz. per long ton. | ·220 oz. per long ton. |
| 11·00 ,, | ·122 ,, |
| 24·50 ,, | ·252 ,, |
| 21·70 ,, | ·234 ,, |
| 35·03 ,, | ·242 ,, |

The fluctuations are well indicated in the subjoined statistics taken from the daily mill record :—

| Concentrates, assay. | Tailings, assay. |
|---|---|
| 21·20 oz. per long ton. | ·200 oz. per long ton. |
| 22·80 ,, | ·200 ,, |
| 24·20 ,, | ·200 ,, |
| 7·00 ,, | trace ,, |
| 8·40 ,, | ·100 ,, |
| 21·80 ,, | ·200 ,, |
| 16·60 ,, | ·150 ,, |
| 27·80 ,, | ·250 ,, |
| 34·30 ,, | ·250 ,, |
| 66·00 ,, | ·400 ,, |
| 42·40 ,, | ·400 ,, |
| 13·60 ,, | ·100 ,, |
| 40·70 ,, | ·450 ,, |
| 40·20 ,, | ·400 ,, |
| 31·00 ,, | ·250 ,, |
| 35·40 ,, | ·350 ,, |
| 14·04 ,, | ·100 ,, |
| 44·50 ,, | ·500 ,, |
| 19·00 ,, | ·175 ,, |
| 22·20 ,, | ·150 ,, |

Inasmuch as the pulp consists of 60 % slimes, and the whole value is contained in exceedingly fine grains of mispickel, the work done by the Frues must be regarded as marvellously good.

Many of the Transvaal mills employ Frue vanners. At the Ferreira, in 1898, operating on stone assaying 21·931 dwt. per ton, they saved 3246 t., equal to 2·465 %, having a value of 5·086 oz. fine gold per ton, and the escaping tailings assayed 7·083 dwt. or ·354 oz. per ton. The cost of concentration was 9*d.* per ton. At the George Goch, the saving of amalgam in the catch-boxes on the vanners is said to pay for

the cost of vanner treatment, and the final tailings are reduced to an average of 10 gr. per ton. At the Jumpers, Frues have been replaced by double troughs (p. 332).

The only available statistics of cost of Frue vanner concentration are given below:—

FRUE VANNERS: WORKING COST.

| Name. | Stps. | % Conc. | Cost per ton milled. | | | | | Remarks. |
|---|---|---|---|---|---|---|---|---|
| | | | Labour. | Power. | Stores. | Maintenance. | Total. | |
| Alaska: | | | *d.* | *d.* | *d.* | *d.* | *d.* | |
| Treadwell | 240 | .. | .. | .. | .. | .. | 2½ | |
| California: | | | | | | | | |
| North Star | 40 | 3½ | 5 | 1¼* | ½ | .. | 6¾ | *Water |
| New South Wales: | | | | | | | | |
| Lucknow | 30 | 2½ | 4½ | 1½† | ¼ | 1¾ | 8 | †Steam |
| Transvaal: | | | | | | | | |
| Chimes | .. | 1 | .. | .. | .. | .. | 8 | |
| Ferreira | 80 | 2½ | .. | .. | .. | .. | 9 | |
| Langlaagte | .. | ½ | .. | .. | .. | .. | 4¾ | |
| Robinson | 140 | 2¾ | .. | .. | .. | .. | 4¾ | |
| Wemmer | .. | 2½ | .. | .. | .. | .. | 11½ | |
| Worcester | .. | 1¾ | .. | .. | .. | .. | 7½ | |
| Victoria: | | | | | | | | |
| S. German | .. | .. | 4¼ | .. | .. | .. | .. | |

*Gilpin County Table.*—This machine is an adaptation of the Rittinger bumping table. It is a much simpler and less costly machine than the Frue vanner, but in capacity and efficiency it does not compare at all. It is favoured in Colorado, and a few are now employed in the Homestake mills, S. Dakota, where the concentrate is bulky and of very low grade. It is quite unfit to deal with fine-grained rich mineral. The vibration or bump is applied endwise.

*Halley's Table.*—An Australian imitation of the end-percussion Rittinger table, this has the great disadvantage pertaining to non-automatic discharge of concentrates. It is shown in

Fig. 99, and is mainly an inclined table 8 ft. by 4 ft., slung at the corners, the inclination being adjustable. A 3-armed cam rotating against a spring produces the bumping. It is costly to run, and of little use on fine pulp.

*Hendy-Norbom Vanner.*—Belonging to the endless belt variety, this concentrator is built entirely of iron, the shaking frame or belt-carrying table being of heavy channel irons. The eccentrics on the driving shaft are adjustable, so that the table can be given different degrees of stroke, 1 in., 1½ in. and 2 in. being mostly used. Either canvas or rubber belts can be adopted. The mechanism which controls the motion of

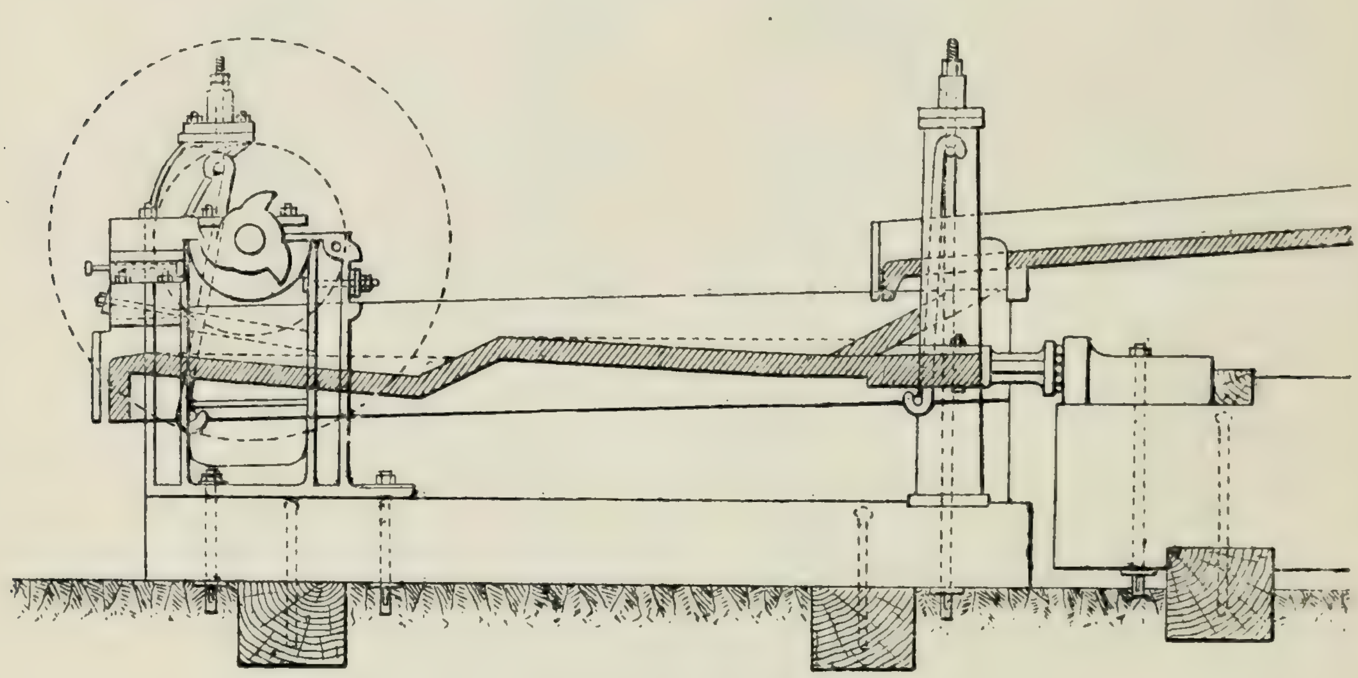


FIG 99.—HALLEY TABLE.

the table consists of four curved springs, upon which the shaking frame rests, two of these springs being placed on each side of the frame, but curved or bent in opposite directions. The driving shaft is fitted on one side, and the eccentrics on this shaft, when imparting a side motion to the table, cause the two springs on one side to bend downward and the two opposite springs to straighten upward, which gives to the table a peculiar, gentle, side-tilting movement. The longer the stroke the greater the tilting. It therefore follows that this concentrator can be operated on two different principles: (*a*) with a rubber belt, giving a quick side-shaking motion and short stroke, producing a slight, gentle tilting or dipping of

the edges of the belt: (*b*) using a canvas belt, giving a slow side-shaking motion and long stroke, producing an increased tilting of the table. It is claimed that this double adjustability enables the machine to adapt itself to a greater variety of gold-bearing ores. It is also claimed that it permits of a very close saving, for the reason that the combination of the side-shaking with the tilting motion obviates the tendency of the pulp to pile up along the edges of the belt. This constitutes the essential feature of the machine, and, if effectually accomplished, it is a decided improvement. No record of its work, however, is obtainable. It is made by Hendy, San Francisco.

*Johnston Vanner.*—This possesses the same feature as the foregoing, and is made by the Risdon Iron Works, San Francisco.

*Triumph Vanner.*—Closely resembling the Frue, and owned by the same firm, this endless-belt machine may be found in a good many Californian mills. At the North Star are 12 of them, alongside Frue vanners working on the same ore, so that comparison is easy. In the words of P. R. Robert,* manager, "the Triumph saves the sulphurets very well; but to do so, it requires constant attention and adjustment, whereas the Frue needs very little attention, is automatic in its action, and delivers the concentrates in a regular manner." The machines are run at a slope of $\frac{1}{4}$ in. per ft., and travel 26 in. and vibrate 230 times in a minute. The film of ore on the belt may only be half of that on a Frue; repairs cost about one-third, or under 10*s.* a year per machine.

*Wilfley Table.*—Amongst the inventions of recent date for the separation of finely crushed ore, this calls for more particular notice. The rapidity with which its introduction in large numbers into the mining fields of every quarter of the globe has taken place, bears witness to the fact that it came to supply satisfactorily a long-felt want. Its utility is not confined to pulp of any particular degree of fineness; provided always that efficient hydraulic classifiers are introduced before the material is admitted to the table, anything can be

* Trans. Inst. M. & M., v. 155 (1897).

treated, from coarse sand down to slimes of almost impalpable fineness.

A close examination of the claims to novelty on which this invention is based discloses much ingenuity in combining a large number of important modifications and adaptations (based primarily upon apparatus already well known) in such a manner that their advantages have been retained, while what was useless or detrimental has been discarded. For example, bumping tables of different kinds had long been in use, and so had vanners and others with a side shake. In the Wilfley, the motion is rapid at one end of the stroke and slow at the other, preserving all the advantages of a bump or jerk while the action is absolutely smooth and noiseless. Again, tables with riffled surfaces had been in use; so also had plane surfaces both in the form of tables and travelling belts. It remained for this machine to combine a riffled and a plane surface while preserving the advantages of both.

Essentially the Wilfley concentrator consists of a table about 7 ft. wide and 16 ft. long, constructed of laths planed truly flat and covered with a smooth waterproof material. Upon this a number of taper riffles of varying lengths are secured, in such a way that the driving end, where the pulp is fed on, is practically covered by the riffles, whereas the concentrates end is smooth and free from riffles. The table rests upon rollers, the carriages for which are secured to cross timbers, hinged in such a way that by the simple movement of a lever the angle of inclination towards the tailings side can be altered at pleasure. The whole is mounted on a solid timber foundation, on which is bolted the motion bedplate, with its crank-shaft, driving pulleys and toggle gearing.

As soon as the pulp is fed on to the table, the rush of water tends to carry it across the table in the direction of the slope towards the tailings side. The riffles, however, catch all the heavier particles and, aided by the reciprocating motion, compel them to travel in the direction of the length of the table. As the riffles taper and become shallower, the upper surface of the pulp, which is barren material, gets washed over them gradually, and falls into the tailings launder, while

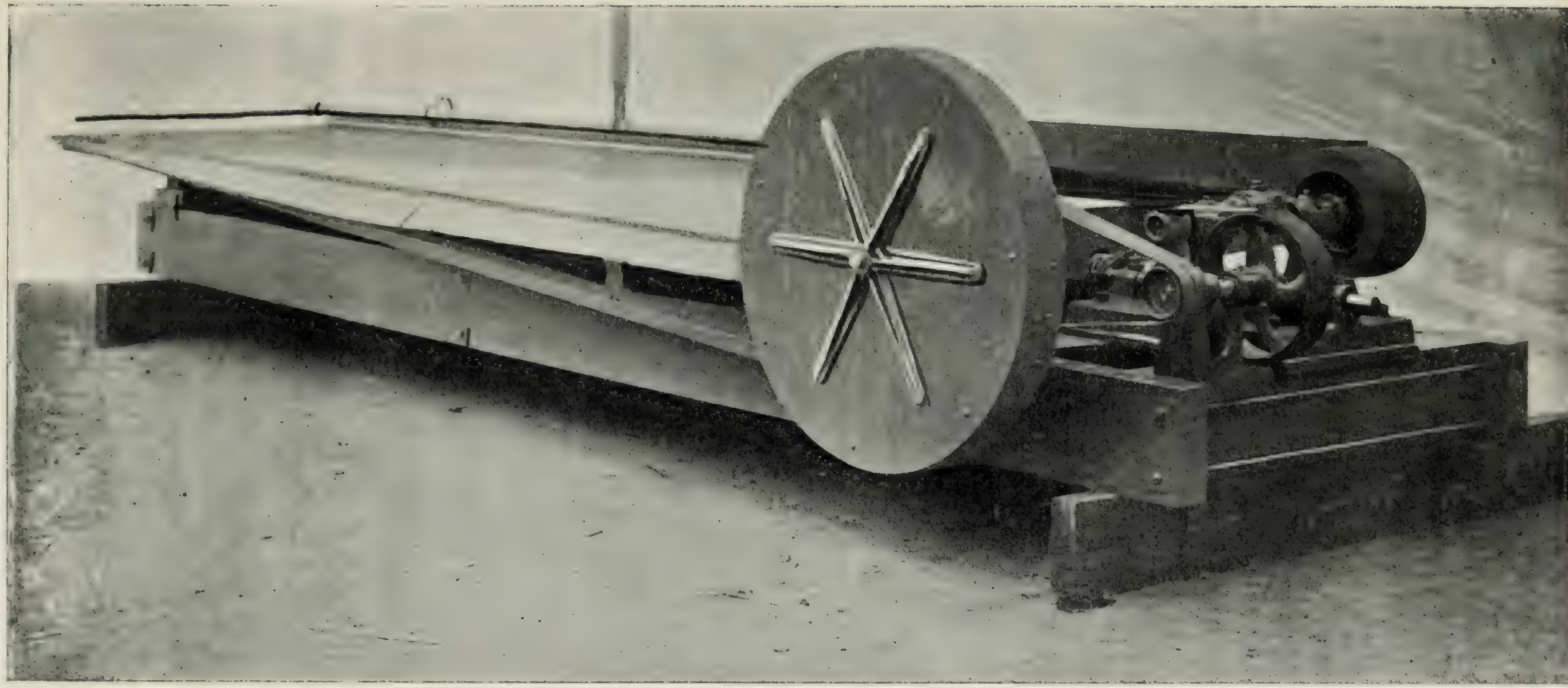


FIG. 100.—WILFLEY CONCENTRATOR.

the heavier particles, which have settled down, move forward and out on to the smooth portion of the table. Here the continued flow of a thin surface of water over them clears them further from gangue, and causes them to range into streaks of the component minerals according to their specific gravity. The heaviest naturally resist the flow of water most, and come off on the higher part of the slightly inclined concentrates lip, and so on in gradation down to the lower corner. At this point, where the junction between the mineral streaks and the gangue occurs, a certain amount of middlings is found, and provision is made for carrying this down an inclined shoot to a raff-wheel elevator, by which it is automatically returned to the head of the table, to be re-treated.

Very high grade concentrates and the poorest possible tailings are the result, while at the same time a much larger amount of work is done in a given time than by other concentrators. From 15 to 30 t. per 24 hours can be satisfactorily treated at an expenditure of only 1 h.p. actual.

Further, it is feasible in one operation to cut out minerals of different specific gravities without re-treatment.

At several mines in Western Australia these tables are doing excellent work on free-milling ores assaying 15 to 20 dwt. gold per ton, raising this into a concentrate giving 25 to 30 oz. to the ton, in which condition it is found profitable to ship it to the smelters, avoiding the cost and trouble of any further treatment on the mine.

The sole makers in England are Bowes Scott & Western. A photograph of the machine is given in Fig. 100.

*Woodbury Vanner.*—Differs from the Frue essentially in having the belt divided up into as many as 13 longitudinal partitions. It does much less clean work. Some few examples are working, both in Australia and in California.

**Oil Concentrators.**—Some stir was made recently in milling circles by extravagant claims urged for the advantages of concentration in a medium of mineral oil, and remarkable results were announced from trial runs on Welsh ores. An investigation of the process, known as Elmore's, was made by C. M.

Rolker, at Glasdir, Wales, and the following notes are condensed from his paper * on the subject.

The process is based on an affinity between mineral oil and the pyritic constituents of an ore, whereby it is possible, from a mixture of crushed ore, gangue, water, and oil to bring about a separation of oil loaded with pyritic matter (which floats), and water filled with barren matter (which sinks). The oil used is the thick tarry "residuum" obtained in distilling mineral oil.

The mixture of pulverised ore and water is fed into the end of a drum, which slowly rotates; oil is added by a separate pipe, made telescopic to admit of discharge at different distances from the end. At each end is a circular opening, and within the drum are annular helical ribs, the spaces between them being divided into cells by a number of equidistant blades, to effect a thorough intermingling of the oil with the pulp and water. All three materials are carried to the opposite end of the drum from where they entered, and are discharged through circumferential holes into the subsidence vessel, where the water and gangue are allowed to subside, the buoyant oil carrying the pyritic ingredients floating on top. The consumption of oil on the Glasdir ore is about 2 gal. per ton.

From the subsidence vessel, the water and gangue are drained at the bottom, while the oil carrying the mineral is pumped to a hydro-extractor, having a solid basket with a projecting flange at the top, running at 750 to 800 rev. per minute. During the action of the machine, the mineral and water pack in vertical walls; the oil, floating on the water, rises vertically up the wall, creeps over the flange or lip of the solid basket, drops into the space between the basket and the enclosing shield, and from thence goes to temporary storage tanks, whence it is pumped again to the reservoir or stock tank. The separated water is let out through an opening at the bottom, covered with a cone cover, after the machine comes to rest. The mineral is packed along the inner periphery of the basket. The final draining of the oil and water is

* Trans. Inst. M. & M., viii. 379 (1900).

accomplished in a smaller hydro-extractor, with a perforated basket or periphery. The oil and water escape through the meshes; and concentrates, carrying about 4 % oil and 4 % water, remain behind.

On the low-grade ore from the Glasdir mine, carrying on an average about 1·12 % copper, and ·049 oz. gold and ·8 oz. silver per ton of 2240 lb., Rolker got a saving of 69 % of the gold, 65 % of the silver, and 70 % of the copper, with a rate of concentration of about 14 into 1.

If the viscosity of the oil requires to be increased, the inventor adds 2 to 3 % of "mineral butter," a later distillation product; if, owing to climatic conditions, the oil has to be thinned, a previous distillation product is added. The temperature of the oil and water at the Glasdir works is kept between 54° and 57° (? F.), though it is possible that wider margins may be permissible. The drawbacks to the process would seem to be mainly :—

(*a*) In each instance, according to climatic conditions and degree of mineralisation, the viscosity of the oil has to be determined and maintained.

(*b*) The right quantity of oil and water must be ascertained, as well as the point at which the oil is to be introduced.

(*c*) The number of blades in each revolving cylinder, the length of cylinder, the number of times new oil is to be introduced, and the number of cylinders or drums required, may vary in each case.

(*d*) The mineralisation of the feed material must be constant for a determined quantity of oil and water. Sudden increase in mineralisation will cause the oil to sink in the water owing to overloading, and this means loss. The water supply must be under control of the operator at the mixer, and the machine must be constantly watched.

(*e*) In a case where the gold runs more with the iron than with the copper pyrites, as copper pyrites seems (according to the inventor) to adhere to the oil quicker than iron pyrites, concentrates richer in copper may be produced at the expense of tailings running higher in gold.

(*f*) An excess of oil will take up rocky particles and bring about poor concentration.

(*g*) Ore or tailings which have been exposed to oxidation or tarnishing must be re-ground to brighten them before the oil will take them up. The process is not applicable to oxides or earthy ores.

On the whole, Rolker regards it as "a process requiring close attention and intelligent treatment, mainly applicable to freshly-produced tailings or copper and iron pyrites ores carrying precious metals, as these will likely offer a uniform grade of mineralisation. To individual mines with low-grade pyritic ore the process will also apply. The advantages are a smaller consumption of water and a higher yield than in wet concentration, with ores similar to the Glasdir ore. It is very difficult to estimate the cost of the process, but, roughly speaking, it can be taken at the usual cost of concentrating such ore, plus 2 gal. of oil per ton of ore, plus the royalty the owners may charge. All extra cost of oil loss (price at Barmouth, Wales, $8\frac{1}{8}d.$ per gal.) and royalty to be paid has to be covered by the difference in yield between wet concentration and this process." It hardly promises to effect a revolution in concentrating methods where gold is involved.*

**Dry Concentration.**—For many years efforts have been made to accomplish concentration by the help of air instead of water, but the results attained have been far from satisfactory, though several pneumatic concentrators or classifiers have been put on the market, both in this country and in the United States.

That bearing the name of Waugh and Bignell is said to be operating successfully in the Black Hills, S. Dakota, presumably on the refractory Potsdam ores, in preparation for roasting and chlorination.

Clarkson and Stanfield's machine is in use on one of the Welsh gold mines, but has failed to find general favour. In the author's experience, the classification effected by it is not sufficiently precise and definite for practical utility.

Somewhat better results seem to have been secured with the Mumford and Moodie separator. This has been employed

* From independent experiments it would seem that if the gangue particles possess considerable lustre they are largely taken up by the oil.

in W. Australia and in Canada, and its working is described by W. McNeill* as efficient. At Hannan's Brownhill, the dry-crushed ore passing a 25- or 30-mesh screen is elevated to the separator, which divides it into about equal proportions of "sands" and "slimes." The arrangement is shown in Fig. 101 : the ore arriving by the bucket-elevator *a* is fed immediately into the hopper *b*, no material drawback arising from the intermittent supply, because of the equalisation produced by the length of shoot *c* and the action of a little baffle-disc attached to vertical spindle *d*. This last, driven by cone-pulley *e* and bevel-gear *f*, imparts a rotary motion to the disc *g* and fan *h*. The crushed ore falling on the rotating disc *g* is thrown off by centrifugal action in a continuous fine shower, and is subjected at the same time to the influence of an air current set up by the fan *h*. This current conveys the lighter particles into the outer casing *i*, where they settle down, and whence they are drawn by the discharge opening *k*. The coarse particles resisting the air currents fall down over the grid *l* and collect in the inner chamber *m*, the issue from which is at *n*. The action of the machine is regulated by altering the speed of rotation and by adjusting the air supply per medium of a damper *o*. Sometimes it is found necessary to add a bumping arrangement, to induce the sands and slimes to descend regularly in their respective compartments, even though the sides be made quite steep.

Compared with revolving or shaking screens, McNeill considers this machine to offer the following advantages :—(*a*) absence of screens liable to choke and needing constant renewal ; (*b*) easy adjustment and control of size of particles carried over ; (*c*) the same air current being in continuous circulation inside the machine, no dust escapes. On the other hand, the terms "sands" and "slimes" used by him do not accurately define the gauge of the two products, and it is certain that to secure 50 % of sharp clean sands the other 50 % will contain much gritty matter which cannot be regarded as slimes. This fact has been emphasised at Deloro (Canada), where pulp from ball-mills and Niagara pulverisers was treated

* Trans. Inst. M. & M., vi. 250 (1898).

by this machine. The result was 60 to 70 % of sands and 30 to 40 % of slimes ; but it was found that the latter contained much leachable material, and, in passing it through V boxes

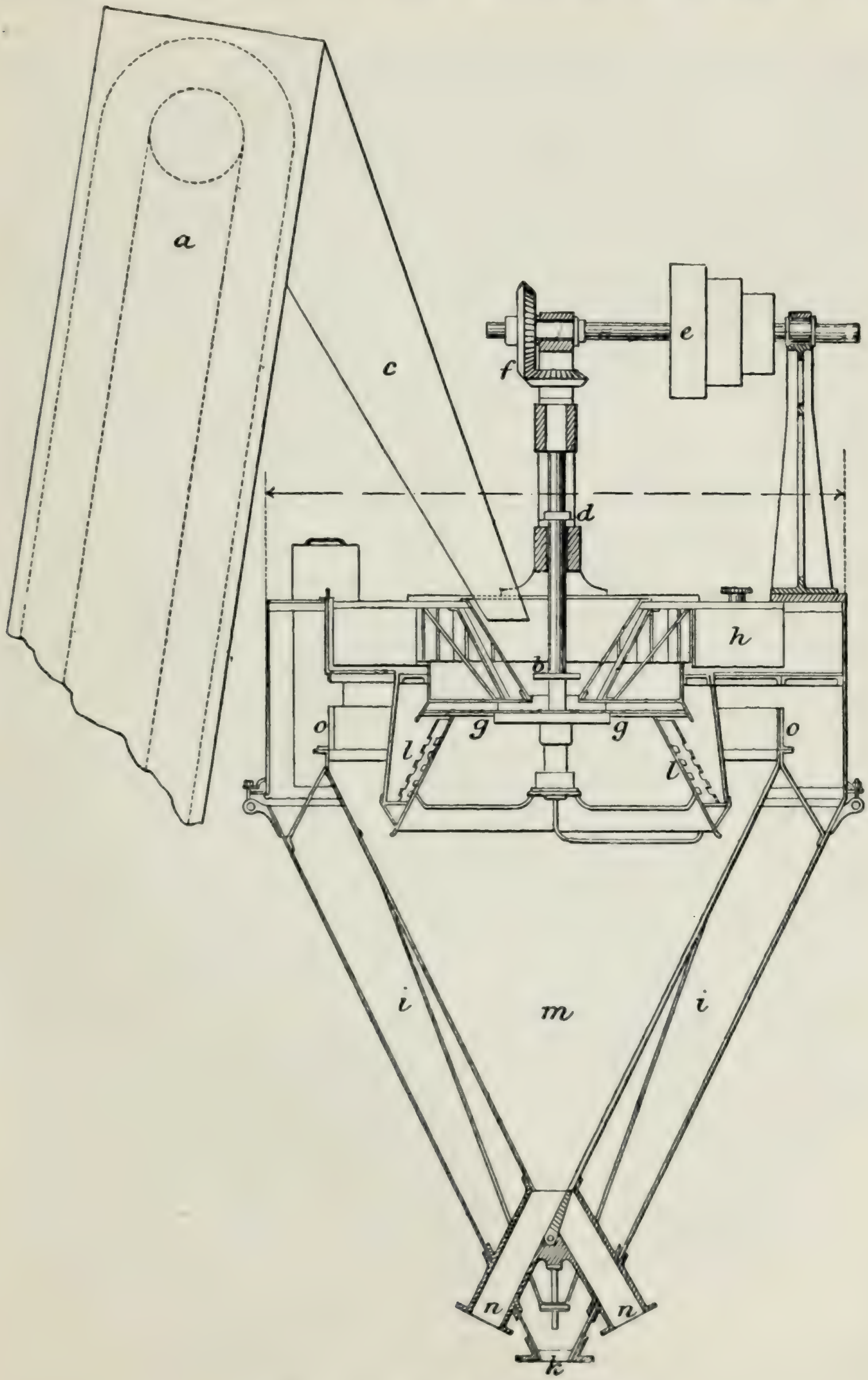



FIG. 101.—DRY CONCENTRATOR.

no less than 60 % of it was taken out in a condition that readily admitted of percolation. This sufficiently proves the shortcomings of dry classification.

The machine is made in several sizes, ranging from 8 ft. high and 3 ft. 6 in. diam. to 13 ft. 9 in. high and 7 ft. diam., the power required being ½ to 3 n.h.p. The large size has a capacity of about 30 t. per diem. It is sold by the Cyanide Plant Supply Co.

**Treatment of Concentrates.** — The sulphurets as collected from the concentrate boxes, or from the troughs in which blankets are washed, are thoroughly saturated with water and in a condition of mud. The first step is to get them into a dry state, whether they are to be roasted and chlorinated on the spot, or to be disposed of to smelters at a distance.

At Lucknow, the drying is accomplished in the following manner. One corner of the battery building, as seen in Fig. 102, is set apart for the purpose. Hither the concentrates are trucked, as mentioned on p. 341, the rails lying just beyond the heads of the vanners. Below the rails about 3 ft. are a series of 4 sheet-iron pans, about 10 ft. sq. and 1 ft. deep, supported on low masonry walls, running longitudinally, and built up from a concrete cement-lined bay, with inclined bottom, so that all moisture condensing in it will drain away. Into the space beneath the pans is carried the exhaust steam from the battery engine. The pans are thus furnished with abundant heat at no cost. Each day's yield of wet concentrates is tipped into one pan of the series. On the second day it is turned over and broken up, and perhaps transferred to another pan, but this labour and its contingent dust are avoided if possible. When dry, the concentrates are shovelled out on to the sheet-iron platform alongside, thoroughly mixed together, all knobs of caked mineral being at the same time bruised into powder by the back of the shovel, and fed into the hopper shown on the left of the filled bags. This hopper carries an automatic sampler, simply a sectionised tube (p. 177) 2 in. diam., which delivers

$\frac{1}{16}$ of the bulk into a bucket and the balance into a controllable shoot opening into the mouth of the bag held below it. The bag stands on a platform scale set at 85 lb., being

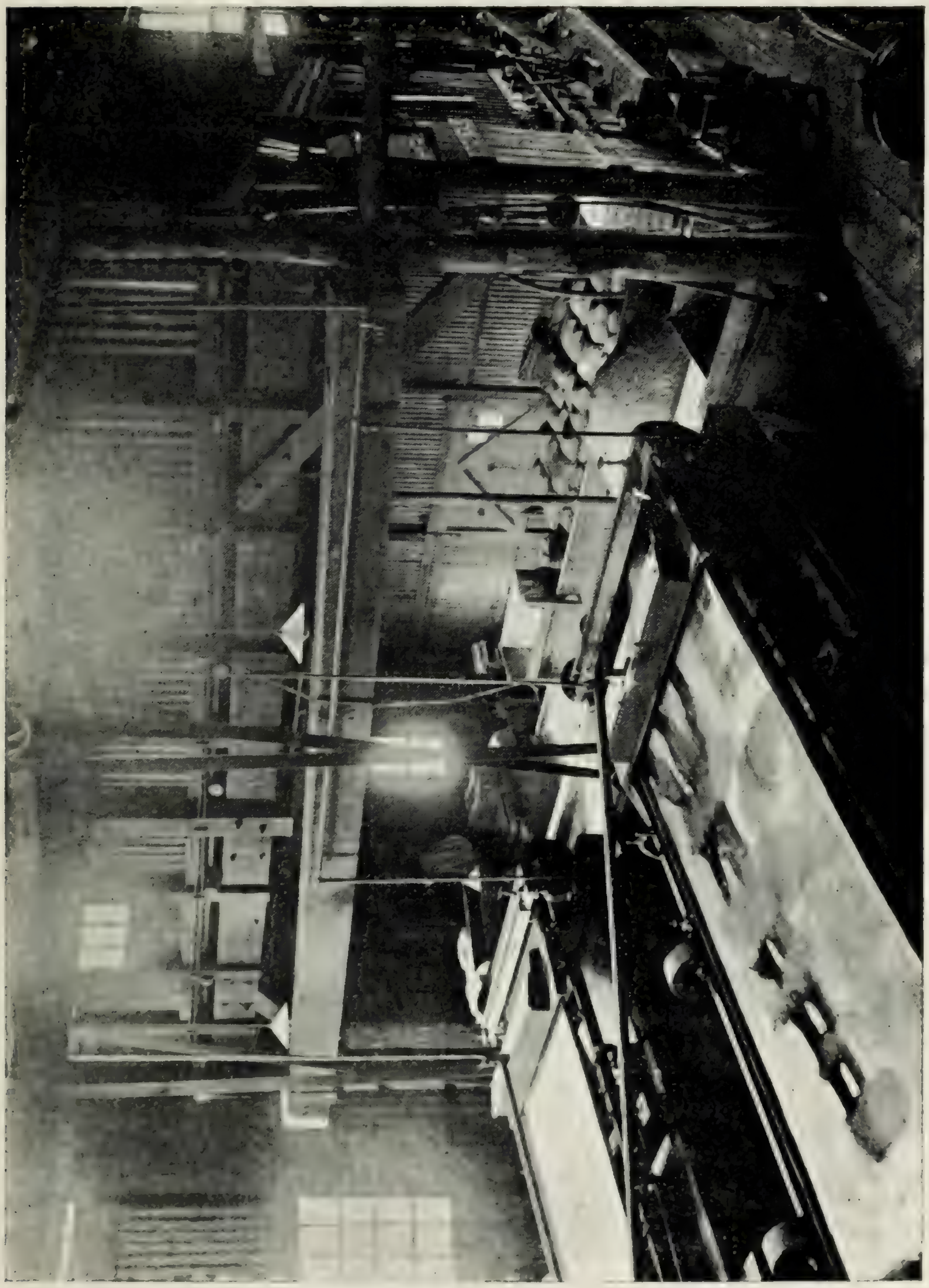


FIG. 102.—VANNERS AND DRYING ROOM.

1 lb. for the bag and $\frac{3}{4}$ cwt. for the ore. This is a convenient weight for handling, and enables the bag to be sewn so that substantial lugs are left, thus avoiding the use of grappling-hooks,

which tear the bags and cause loss. The cost of handling 299 t. in 1899 was 7*s.* 10*d.* per ton for labour, and 5*s.* 8*d.* per ton for stores, chiefly bags. One man at 7*s.* 6*d.* per 8-hour shift does the whole work, averaging just about 1 t. per diem. Single bags are used when despatching only to local smelters; but previously, when the product was shipped to Swansea, either double gunny bags or casks (second-hand sherry pipes) had to be employed, at much higher cost.

The automatic sampler is very reliable when the stuff is carefully broken to dust; otherwise its sample will assay much under the mark, because the caked pellets are far the richest material, and these cannot enter the small division leading to the bucket. The bucket contents are emptied into a clean bag, and taken to the assay office for further reduction, when each 3-ton (long tons) lot (80 bags) is complete. Two 3-ton lots make a truckload, and secure minimum railway rates.

The mines of the Remedios district, Colombia, ship their concentrates to Europe. The dried material is packed in stout holland bags, which are placed in strong wooden boxes; 92 lb. weight goes into each bag, making, with the box, which weighs 20 lb., a parcel of 112 lb. Two of these constitute the *carga* or load of a pack-mule. Sacks and raw-hide bags were tried, but would not stand the rough handling.

At the Ferreira mill, sacking of concentrates and cartage to chlorination works costs only 2½*d.* a ton, the said chlorination works being on the adjacent Robinson mine.

# CHAPTER VIII.

## ROASTING.

TWO distinct kinds of roasting are in vogue for treating gold-bearing ores; they are known respectively as an oxidising roast and as a chloridising roast.

*Oxidising Roast.*—This may be said to have a dual object—physical and chemical. The physical result aimed at is an increase in the porosity of the rock, and in some few instances, notably where cyanide treatment is to follow, this may be the chief desideratum. The chemical change to be brought about is the elimination of sulphur and arsenic, and the conversion of sulphides and sulphates, arsenides and arseniates, into oxides. The double object is accomplished by heat and air in one operation, in properly constructed furnaces, and is variously known as calcining, "sweet" roasting (since all acids are removed), oxidising roasting (because other salts become oxides), or roasting "dead" (alluding to the absence of sparks and the blackened appearance of the mineral when the roasting has been thorough). The physical disruption of the ore is, of course, further aided by the chemical changes taking place in its constituents, and thereby the isolation of the gold particles is facilitated.

In the very great majority of instances—perhaps in all, except where antimony, bismuth, or tellurium occurs in the ore—the gold exists in what is known as the "free" state, that is to say, it is not in chemical combination, but only in extremely intimate mechanical association with the other constituents, sandwiched, as it were, between the grains of pyrites. This is clearly shown by the fact that, when such ores are ground sufficiently fine, the gold can be recovered

from them by the medium of mercury. But the presence of sulphur, arsenic, etc., has a deterrent effect on the action of mercury, and the partially oxidised products, such as sulphates and arseniates, are even more detrimental, not only to amalgamation, but also to chlorination and other methods of gold recovery. Hence the complete transformation into innocuous oxides is generally an essential condition of successful roasting.

Deadness or sweetness of roast is judged by furnacemen on somewhat empirical lines. The principal features indicating it are that the ore emits no sulphurous odour and no sparks on being disturbed; that it will "bank up" with an almost vertical face to the heap; and that it rapidly turns black on cooling. While the absence of any of these conditions is proof positive that the roast is not finished, their presence is not an equal assurance that it is. Quenching some of the ore in water dissolves soluble sulphates if present, and the test then consists in immersing a polished iron rod, which should emerge quite bright; or what is much more reliable, and equally simple, addition of a few drops of barium chloride solution, which will produce a milky appearance in the water if the ore still contains undecomposed sulphates.

In discussing the principles which should guide the operator in conducting an oxidising roast, it is impossible altogether to avoid entering into chemical questions, as they underlie the operation. In the early days of the gold industry of this century, these chemical problems were quite ignored, and many disastrous failures were incurred in consequence.

The principal constituent of almost all ores which require roasting is iron pyrites, either the magnetic form known as pyrrhotite or monosulphide ($FeS$), or the common variety called pyrite, which is a bisulphide ($FeS_2$). If these minerals are subjected to heat without air, the sulphur is volatilised. If air has free access to them during heating, the sulphur burns, passing through the stage of sulphurous anhydride ($SO_2$), and rapidly becoming sulphuric anhydride ($SO_3$). This sulphuric anhydride attacks the iron, and forms ferrous sulphate, and unless the heat be sufficiently great to decom-

pose this sulphate of iron, much of the original sulphur will remain in that form. If the heat be duly raised and prolonged, the final result will be ferric oxide ($Fe_2O_3$); but if the heat be unduly raised or prolonged, some of this will be changed to magnetic oxide ($Fe_3O_4$), which is an objectionable feature if chlorination is to follow, the action of chlorine being much more intense on magnetic oxide than on ferric oxide, and the waste of chlorine correspondingly augmented. Again, excess of heat, in presence of a lack of air, may even result in one atom of the sulphur being removed, and a fluid slag of monosulphide of iron being formed. Either or all of these conditions may be encountered simultaneously in different parts of the same furnace.

Next in importance, if it be not, in some cases, of paramount importance, is arseno-pyrite or mispickel, the arsenical sulphide of iron (FeAsS). The expulsion of arsenic would seem to be a much more complicated matter than the elimination of sulphur, because while it is volatile only in the condition of arsenious acid, it will repeatedly undergo modifications in the furnace, according as the ever-varying atmosphere in contact with it is more sulphurous or more oxygenous. In presence of an excess of oxygen, there is a tendency to form fixed arseniates of the iron and other metals present, and these arseniates are decomposed only at very high temperatures; therefore rapid or intensified oxidation of arsenic is to be avoided.

Much the same remarks apply to antimonial sulphides (stibnite, etc.), which, as antimoniates, are even more troublesome than arseniates.

Copper pyrites become first copper sulphate, and finally a mixture of cuprous and cupric oxides, both soluble in chlorine, so that cupriferous ores are not adapted to chlorination without special treatment.

Lead sulphide (galena) demands caution in controlling the rate of oxidation, by reason of its tendency to produce fusible compounds, whether as sulphate or as silicate (derived from the quartz gangue).

Zinc sulphide (blende) requires a high temperature and

an abundance of air, the sulphate demanding a temperature much in excess of that produced in a pyrites furnace.

The chemistry of the oxidising roast of ordinary mixed sulphides such as usually occur in gold-mill concentrates has been described at length by Prof. Roberts Austen,* who says that the first effect of an increased temperature is that sulphur is distilled off, reducing the sulphides to lower stages of sulphurisation. This sulphur burns in the furnace atmosphere to sulphurous anhydride ($SO_2$), and coming into contact with matter undergoing oxidation, is converted into sulphuric anhydride ($SO_3$). The material of the furnace does not enter into the reaction—as it does in some branches of dry metallurgy—but its presence, as a hot porous mass, has considerable influence. The roasting presents a good instance of chemical equilibrium. As soon as this $SO_2$ reaches a certain tension, the oxidation of the sulphide is arrested (even though excess of oxygen be present), and is not resumed until the action of the draught changes the conditions of the atmosphere of the furnace, when the lower sulphides remaining are slowly oxidised, copper sulphide being converted into sulphate mainly by the intervention of the $SO_3$ formed as indicated. Probably by far the greatest part of the iron sulphide only becomes sulphate for a very brief period, being decomposed into the oxides (mainly ferric), and the sulphur passing off. Any silver sulphide present would have been converted into metallic silver at the outset, were it not for the simultaneous presence of other sulphides, notably those of copper and iron, which enables the silver sulphide to become converted into sulphate. Lead sulphide is also sulphated at this low temperature (about 930° F.). As the heat advances, copper sulphate is split up, and leaves the oxides. If, as in this case, the bases are weak, the $SO_3$ escapes mainly as such ; but when sulphates of stronger bases are decomposed, the $SO_3$ is to a great extent split up into a mixture of sulphurous anhydride and oxygen. The $SO_3$ resulting from decomposition of copper sulphate converts the silver into sulphate, and maintains it as such, much as, at a lower temperature, the copper itself had been preserved as

* Pres. Address, Brit. Assoc., 1891, pp. 592–3.

sulphate by the $SO_3$ generated from the iron sulphide. When only a little of the copper sulphate remains undecomposed, the silver sulphate begins, at about 1290° F., to split up, partly by the direct action of heat alone, and partly by chemical reaction between it and the iron and copper oxides. Lead sulphate remains undecomposed at any temperature realised in the roasting furnace, unless silica intervenes. The elimination of antimony and arsenic is similarly complex. Taking the former as an example, and assuming it to be in the form of sulphide (stibnite)—though this is not by any means always the case—the first step is an escape by distillation. Rapid oxidation of stibnite would be likely to result in production of insoluble antimoniates of other metals, therefore a very low temperature is necessary at starting, and steam as a source of hydrogen, to combine with the sulphur as $H_2S$, is a valuable aid. The antimony takes up oxygen, and escapes as volatile oxide. Arsenic behaves in a similar manner, though some of it is volatilised as sulphide.

Enough has been said to indicate that the efficient roasting of mixed sulphides is not a simple matter. On an industrial scale, perfection cannot be attained, and the main principle kept in view is to let the heating and oxidation be gradual at first, progressing regularly, and terminating at the highest point, which should not be prolonged beyond actual needs. This gives the best general result on the average ores dealt with.

The leading features to be striven for in the conduct of oxidation roasting operations on gold-bearing ores are as follows :—

1. Avoidance of manual labour, making both feed and discharge automatic ; utilising gravity as much as possible for movement of the ore through the furnace ; and reducing the moving parts and motive-power of the furnace to minima.

2. Greatest possible economy of fuel, therefore the maximum attainable utilisation of the combustible contents of the ores themselves.

3. Provision of an abundance of oxygen in the form of heated fresh air, accomplishing this by taking advantage of

the waste heat of the furnace, and not consuming fuel specially for that purpose.

4 Most complete exposure of every particle of ore to the heated air.

5. Simplicity and economy in construction of furnace; fewness of wearing and working parts, which should be easily duplicated ; and general accessibility for repairs.

6. Utilisation of all products of the operation, so far as that is possible.

7. Providing a low temperature at starting, and gradually increasing it to full redness at the finish.

Chemical Aids.—The possibility of aiding and hastening the oxidation roast by chemical means, beyond the mere admission of air to supply oxygen, has not received that amount of attention which it deserves. The average length of time occupied in sweet-roasting a charge is such that an extensive installation of furnaces is needed for a daily output of any great amount. This means heavy outlay in plant and extra cost for renewals, because of the limited capacity. In addition the working expenses are augmented by labour and fuel, which might be economised if the speed of roasting were increased.

The addition of nitrate of soda to the ore was proposed some years ago by Prof. Chapman in the United States, but it did not afford results commensurate with its cost, and the suggestion came to nothing. Whatever the article it must be cheaply and readily obtained and must shorten the roasting very materially to be worth consideration. Such an article has recently been tried on the experimental scale with very marked success, permitting a much lower temperature than usual to be maintained, and effecting in about 20 minutes as complete a roast as is generally accomplished in 5 or 6 hours, such sulphur as remained being in the soluble form, i.e. as sulphates, which can be leached out. Should tests on a working scale prove nearly as satisfactory, very material progress will have been made in the treatment of refractory ores.

*Chloridising Roast.*—In certain cases, something more than oxidation is attempted in the furnace, and a chloridis-

ing roast is aimed at. This occurs particularly where (*a*) the ore contains a valuable proportion of silver, which cannot be recovered by the ordinary chlorine leach which serves for gold ; or (*b*) carries such a quantity of copper, lime, or magnesia as will cause a waste of chlorine during "gassing." The remedy is to roast with salt, which, while much cheaper than chlorine, undergoes decomposition in the furnace with the sulphates of the heavy metals, forming sodium sulphates and various chlorides. Further the action of $SO_3$ on the salt generates free chlorine, which the water, set free from the salt crystals, helps to convert into hydrochloric acid. Many of the chlorides and oxychlorides so formed, as those of antimony, arsenic, copper, iron, and lead, are volatile. The addition of salt is usually made at as late a period in the roasting as possible, when only some undecomposed sulphate is left, the quantity varying from 3 lb. to 90 lb. per ton. Occasionally the salt is mixed and admitted with the ore ; but this entails great waste of salt (much of the chlorine being removed as chloride of sulphur), and heavy loss of gold (mechanically suspended in the escaping volatile chlorides and in the heated vapours), the effect being intensified as the roasting is prolonged.

*Gold Losses in Roasting.*—The common assertion that when roasting gold ores containing tellurium there is a risk of loss of precious metal by volatilisation is controverted by T. K. Rose,* who made experiments showing that "as near as could be ascertained, absolutely no gold was volatilised," but that the compound formed by "heating gold with tellurium at a roasting temperature soaked through the ore and was taken up by almost any furnace bottom," so that it was difficult to avoid loss. In roasting tellurium ores, attempts have been made to recover the volatilised gold in flues, but the experiments Dr. Rose made tended to show that "no gold could be recovered under such circumstances, and that it had to be looked for below and not above." It would seem, however, that Dr. Rose experimented with the clean metals simply, and not with raw ores.

Some trials made at the Boulder Main Reef, Kalgurli,

* Trans. Inst. M. & M., vii. 45–6 (1899).

indicated that if telluride ore is crushed wet with the saline water which has to be used in that locality, and it is afterwards roasted, there is a great loss of gold by "volatilisation," reaching as high as 40 to 50 %. But surely it is not necessary in this case to invoke the aid of tellurium to account for the loss, because it has long been established that serious losses will ensue from roasting gold ores in the presence of salt.

Experimenting on rich telluride ores, Sulman * found that when the readily-melted globules are exposed to an oxidising atmosphere they rapidly become coated with a layer of highly-insoluble tellurous acid, which, though thin, is almost like a glaze of porcelain over the small beads of partially-roasted telluride; and this layer not only prevents complete oxidation but, being insoluble in dilute acids and alkalies, effectively protects any reduced gold from being subsequently dissolved in cyanide. He would therefore avoid oxidation at the first stage of the roast, and rather provide a reducing atmosphere, so as to break up the tellurides and liberate tellurium in an elemental condition, thus avoiding both mechanical loss in tellurous acid vapours and the coating by tellurous acid. After the bulk of the tellurium had been volatilised, an oxidising flame would be admissible.

At Cripple Creek, Colorado, the roasting of tellurides is not considered more difficult than that of ordinary sulphides. Very little tellurium is found in the flue-dust, and 90 % remains in the charge. As the tellurium of the tellurides becomes oxidised in the furnace, it is supposed to combine with the iron oxide being formed, making a tellurite of iron which remains fixed in the ore; and unless care is used, gold particles get coated with this compound of iron and tellurous acid, presenting a serious impediment to the action of cyanide. The roasting of Cripple Creek tellurides is very successfully carried out at Florence (Colo.) by using the "residuum" from refining petroleum, which is burned as a gas, and affords a temperature which can be regulated to a nicety. A Pearce furnace is used, divided into 3 stages, and the heat is kept down till the last stage is reached.

* Trans. Inst. M. & M., vii. 48 (1899).

Considerable experiments in connection with the roasting of tellurides made by Cordner James * have shown that with the water obtainable in Western Australia, crushing wet and roasting the concentrates, the losses vary from 15 to 25 %; whereas, roasting after crushing dry, when common salt and other chlorides are not present, the losses average from 1½ to 2 %, and he has been able to operate on a fairly large scale without exceeding these figures. This probably represents a truly mechanical loss in the particles carried away in suspension by the furnace gases, and it may be ultimately recoverable, at any rate in part, from the flue dust. Authorities are agreed that absolute immunity from loss by dusting is impossible of attainment.

From results quoted by C. A. Stetefeldt † on an ore consisting of hard white quartz carrying 7 % calcite, ·65 oz. gold per ton, and a little pyrites, it would seem that possibly chlorination of the gold takes place, since this ore lost 68 to 85 % of its gold when roasted with 5 % salt, but suffered no loss at all in an oxidising roast. All the salt was volatilised. Doubling the quantity of salt did not increase the loss of gold.

Prof. Christy has proved that gold is volatile in the presence of chlorine, at all temperatures from that of boiling water to a white heat, and that the amount of loss depends upon time, heat, surface exposure, and quantity of chlorine. Gold thus volatilised, at least at the temperature of roasting, cannot be recovered by simple condensation of the fumes; it is necessary to employ some agent which will decompose the volatile gold chloride. It appears that sulphur dioxide in the presence of water does this, just as it decomposes the ordinary gold trichloride dissolved in water, precipitating the gold in the metallic state. In a long furnace, with several hearths, the salt is added on the finishing hearth; at the same time the ore on the next hearth is burning and giving off fumes of sulphur dioxide; hence it is only necessary to draw the smoke from the furnace through a sufficiently high coke tower, the coke being constantly wetted by a small stream of water,

* Trans. Inst. M. & M., vii. 50 (1899).

† Trans. Am. Inst. Min. Engs., xiv. 339 (1885).

when the gold fume will be decomposed, and the metal be precipitated on and among the coke, or remain in part suspended in the water, which would be filtered before running to waste. The same water may be elevated and repassed until it becomes too acid. Charcoal may be used instead of coke, and Aaron suggests that probably quartz would also answer. The coke or charcoal has to be burned in order to recover the gold, but quartz would only require washing. It would be necessary to force the draught by suction. For this purpose a fan or blower, made of wood, and painted with asphaltum and coal tar, may be used ; or a steam jet would probably answer, and the steam, condensing partly in the tower, would do no harm at least.

A loss of gold may also occur through too rapid roasting, the heat being too high, and the fine particles of metal being carried away by the violent evolution of the gases of combustion.

**Furnaces.**—Considering the great variety of gold-bearing ores that have to be dealt with, it is hardly likely that one form of furnace will be found to meet all requirements with a maximum degree of efficiency. For instance, though fine crushing would much hasten oxidation, it is not always admissible, on account of subsequent treatment. Then the percentage of combustible matter (sulphur, arsenic, etc.) in the ore must have an important bearing on the form of the furnace. Again, the quantity of material to be dealt with may have to be considered : few mills produce sufficient concentrates to fully occupy any mechanical furnace, and this fact seems to have helped prolong the survival of the hand-worked reverberatory in many places, although that would seem to be a poor excuse for such costly manipulation as it entails.

Speaking generally, the author is inclined to think that the several kinds of furnace will have their most useful spheres defined by the following conditions :—

For finely-ground (40 mesh and finer) clean sulphurets, in which absolute sweetness is not essential, a shaft furnace.

For coarsely-ground (less than 40 mesh) sulphurets, and for only moderately sulphuretted ores, a revolving cylinder.

For any ore which requires an absolutely sweet roast, a mechanical rabbler with fixed hearth.

The slovenly and noxious method of calcination known as "pile roasting," which dispenses with a furnace of any form, though used extensively for the desulphurisation of copper pyrites to make it ready for the smelter, demands no description here, as it is quite unsuitable. Even in the very few instances where gold mines afford a massive pyrite, the exigencies of subsequent treatment demand a more scientifically conducted roast than is possible under the crude conditions afforded by calcining in huge heaps. Incomplete oxidation on the one hand and excessive sintering on the other, while tolerable when smelting is to follow, are inadmissible for leaching processes. Even as a mere preliminary to a well-controlled calcination, there seems to be nothing gained by the double handling, especially as the second ignition would be much more difficult than the first. The cost of roasting lies not in the burning of the bulk of the sulphur—that, in fact, will maintain its own combustion—but in the removal of the last ½ or ¼ %. Moreover, at the great majority of mines the roasting has to be done on very fine-grained sulphurets gathered from the concentrators, most unlikely material for burning in heaps.

The object of all furnaces is to promote the access of heat and oxygen to the ore, hence their differences of construction depend in the main upon the means which are adopted to favour these ends. They may be first divided into two principal groups, according as the ore is worked over, so as to expose new surfaces to the air and heat: (*a*) by manual labour; and (*b*) by mechanical contrivances.

*Manual Furnaces* are represented by the various forms of reverberatory calciner, which have stood the test of time, and are still considered by many to be the most efficient furnace for securing a sweet roast.

The essential features of the common reverberatory are a long, broad hearth, having a gentle inclination (say, 1 in. per ft.)

from the fire-end up towards the feed-end, and covered by a low vaulted roof. The height of the hearth above the ground-line should be about 3½ ft., and its width about 12 to 15 ft., for most convenient operation. The length varies greatly—60 to 80 ft. is desirable from the standpoints of fuel economy and efficiency—but these considerations are often overlooked or made subservient to economy in cost of construction, the length being reduced to 40 or even 30 ft. Sometimes the hearth is divided laterally into 3 or 4 sections by drops of a few inches each, but more generally the bed is of one uniform grade without divisions, though still called "3-hearth" or "4-hearth," which terms, however, simply indicate that the length is 3 or 4 times the width. The hearth should be built of brick set on edge, as close as possible, in fireclay. For the lowest section of the furnace, as well as for the fireplace and bridge, best firebrick will prove cheapest in the end. The upper sections of both hearth and vault may be of sound ordinary brick. The space beneath the hearth should be made solid with rubble, to conserve the heat. The thickness of the vaulted roof is generally one brick (9 in.), and the distance between hearth and vault commences at about 2 ft. at the fire-end (less in a small furnace) and diminishes somewhat towards the other.

It is essential that the fuel used be of a long-flamed character. For this reason, wood is very generally preferred, though often much more costly than coal. Probably at no distant date wood will cease to be available in many places, and there will be a tendency to instal gas-producers and take advantage of the cheapest fuel that can be had, as is already under consideration at Mount Morgan and elsewhere.

The pulverised ore is fed on to the uppermost section of the hearth, and spread so as to lie 2 or 3 in. deep (about 15 lb. per sq. ft.), in a succession of ridges and furrows. Spreading is performed by heavy iron "paddles" or rabbling tools, which are admitted by small "working doors," in cast-iron frames 6 to 9 in. high and tapering from 12 in. long on the outside to 18 in. on the inside. A series of these doors on each side of the hearth for its whole length, at intervals of about 6 ft., permits the charge to be periodically rabbled and moved on

towards the fire. The working doors should alternate on the opposite sides. Long flues for condensation of the sulphurous and other vapours generated are necessary, and a chimney to give draught to the fire. The products of combustion from the fire, together with air entering at the fireplace and at the working doors, pass over the exposed surfaces of ore, and produce the desired oxidation. The oxidised ore escapes through a slot in the hearth just behind the firebridge.

A form of reverberatory favoured in Victoria is shown in Fig. 103 :—*a*, air pipes above the fire ; *b*, fireplace ; *c*, firebridge ; *d*, discharge slot for roasted ore ; *e*, hearth ; *f*, working doors ; *g*, charging hopper ; *h*, flue leading to condenser pocket *i* ;

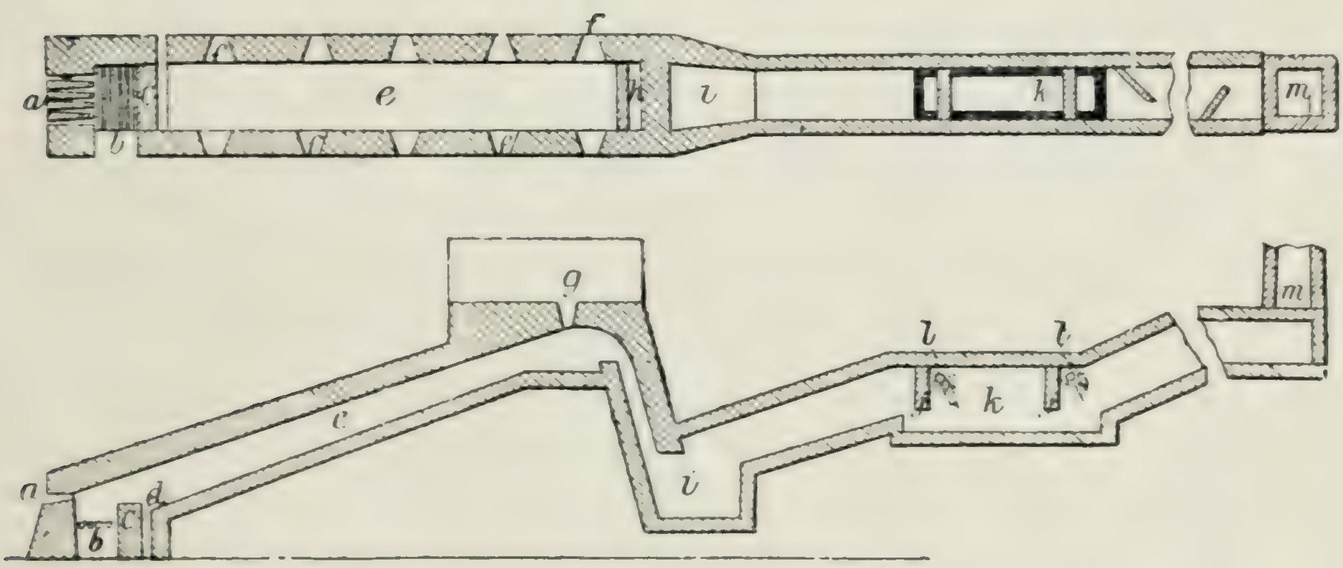



Fig. 103.—Manual Reverberatory Furnace.

*j*, dampers for checking flow of gases ; *k*, leaden cistern in which gases condense, aided by sprays *l* ; *m*, chimney. It is highly efficient, pyrites containing about 7 % arsenic and 6½ % sulphur being roasted sweet in 12 to 18 hours. But the cost is very high. Treating 4½ to 5 t. per 24 hours, the cost is about 11*s*. 7*d*. a ton for labour and 8*s*. for fuel ; total 19*s*. 7*d*. Cosmo Newbery puts the average cost of roasting by reverberatory throughout Victoria at 16*s*. a ton, and figures which the author collected in California in 1894 show much the same cost there. T. A. Rickard quotes 18*s*. 9*d*. a ton at Clunes, Victoria, in a furnace 40 ft. by 5 ft. ; and 29*s*. 9*d*. per ton at Charters Towers, Queensland. At Mount Morgan, with wood at 15*s*. 2*d*. per ton, it costs 17*s*. 10*d*. per ton to

reduce the sulphur in pyritic ore down to ·09 %. This, of course, is simply throwing money away. Some economy of operative cost has been aimed at by building the furnaces with superposed hearths, but the increased cost of construction and expense for repairs seem to have more than counterbalanced the saving.

A good Californian example of the hand-worked reverberatory furnace has a hearth 80 ft. long by 12 ft. wide, inclined from fireplace to flue without any break. It has 12 working doors in the side, and will take 3 charges of 2400 to 2600 lb. in 24 hours. The distribution of the charges on this single hearth is maintained as carefully as if there were 3 distinct hearths. The section nearest the flue, where the ore is quite cool, is little more than a drier. In the middle section, the greater part of the sulphur is burned off, and here the ore is spread very thin, and is rabbled constantly. The section nearest the fireplace is kept at the highest temperature possible short of sintering the ore, and here the dead roast is accomplished. In moving the charges from one section to another, they are rigidly kept distinct. A very low heat suffices in the first section, the ore looking black on the surface in daylight, and dull red at night; but little rabbling is done at this stage.

An Australian type of manual furnace, advocated by T. G. Davey,* resembles that described by Küstel, the hearths being made level, and separated by drops of about 2 in., to serve as a guide to the workmen, and prevent the possibility of ore at different stages of roasting becoming mixed. In Fig. 104 (scale $\frac{1}{8}$ in. = 1 ft.), *a* is the finishing hearth, *b* the middle hearth, *c* the cold hearth; each is 9 ft. sq., with rounded corners. The rabbling doors *d* permit most efficient raking. The hearths and crown are put in with dry rubble, and afterwards grouted with fireclay. The spring of the crown is reduced to a minimum compatible with stability, so as to secure even distribution of the furnace flame over the whole area. The crown is 28 in. above the hearth at the firebridge, and falls to 20 in. at the first step, which figure is maintained at successive steps.

An example of roasting (for chlorination) of richly arsenical

* Trans. Inst. M. & M., viii. 474 (1900).

concentrates (see p. 321) which have passed through a 35-mesh brass wire screen (producing much slime), in Brazil, is recorded

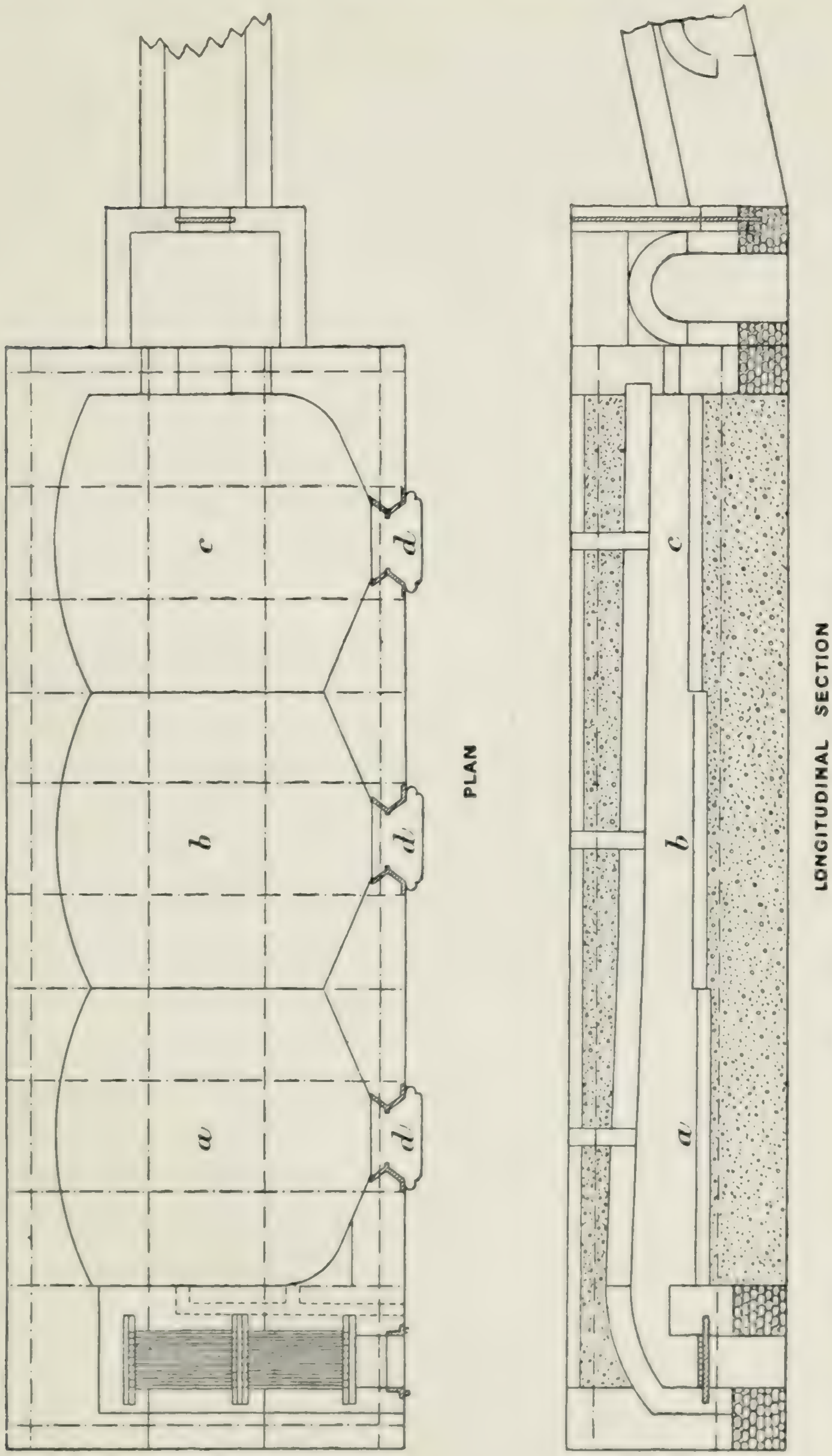



FIG. 104.—MANUAL REVERBERATORY FURNACE.

by Spencer Cragoe,* who used a 2-hearth reverberatory by Fraser & Chalmers, with a nominal capacity of 5 short tons,

* Trans. Inst. M. & M., viii. 123 (1900).

and an actual daily output in practice of 4·3 long tons, which is much the same thing. The fuel is cordwood "of fair calorific value." Labour is on contract, at the rate of 8*s.* 8*d.* for every 5 cars raw concentrates; each car holds about 650 lb., and the daily charge is 15 cars, or about 5 t. The contract includes tramming from mill to furnace, drying if necessary, charging, roasting and drawing. The raw concentrates assay about 2·5 oz., and lose 38 % of their bulk in roasting. Towards the finish of the roast some care is necessary to avoid sintering. "Sweetness" is checked by the ferricyanide test. The cost per ton, on the basis of 140 t. monthly, is as follows:—

| | *s.* | *d.* | % |
|---|---|---|---|
| Reduction officer, ¼ time, at 30*l.* p. m. | 1 | 0 | 7·00 |
| Labour, 6 contractors and 3 boys | 5 | 7 | 36·39 |
| Fuel, ·56 cord of wood at 16*s.* 8*d.* | 8 | 8 | 56·61 |
| Total cost per ton | 15 | 3 | 100·00 |

At the great Treadwell mill, Alaska, the old-fashioned reverberatory furnace has had the distinction of displacing two types of mechanical furnace, viz. the Spence and the Brückner. The work there consisted in sweet-roasting concentrates for chlorination, but this has now been discontinued, and the concentrates are sold to a smelter. The cost in 1898, on 7700 t., was 12*s.* per ton for labour, and 5*s.* per ton for fuel, wood being about 17*s.* a cord. The concentrates carry 40 % sulphur.

A double reverberatory at Haile, S. Carolina, furnishes 2 t. of roasted ore per 24 hours, with an average consumption of 1 cord of wood at 6*s.* 3*d.*, and the employment of 4 labourers. The cost per ton of roasted ore is 10*s.* 11¼*d.*

The effectiveness of the hand-worked reverberatory is gained at the sacrifice of fuel and labour. The oxygen carried in by the air entering at the fireplace, even when supplemented by air pipes, as shown in Fig. 103, is mostly consumed in supporting the combustion of the fire, leaving but little for effecting oxidation of the ore. The demands for the latter object are supplied by the air admitted at the working doors, which, however, is cold, and therefore lowers the tem-

perature of the furnace, and causes waste of fuel. If the work is hastened, not only is additional fuel consumed, but considerable mechanical loss of ore may be caused by the excessive draught. If the work is prolonged, the cost of labour is augmented. Therefore it does not seem that any great improvement can be introduced in the manual reverberatory system, unless it be in the direction of utilising the waste heat for warming the fresh-air supplies, which, however, does not promise to be very feasible.

Only a limited number of mines produce sufficient sulphides to afford constant work for one of the modern mechanical furnaces, and this is a principal reason for the retention of the reverberatory; but surely a more rational proceeding is for adjoining mines to combine in this respect, as in ordinary milling, or to sell the product to a central ore reduction works.

*Mechanical Furnaces* comprise four distinct classes:—(1) shaft furnaces; (2) rotating cylinders; (3) mechanical rabblers; and (4) rotating beds.

(1) Shaft furnaces have come into very limited use in calcining auriferous ores, despite the acknowledged advantages of dropping the ore in an ascending stream of hot air, which is sometimes sought to be attained, though very imperfectly, by making deep steps between the several hearths of the reverberatory.

The well-known Stetefeldt furnace is an early example of the shaft furnace, the ore falling without let or hindrance straight from the top to the bottom of a 30- or 40-ft. shaft; this obviously does not permit of a sufficiently prolonged contact between ore and heated air to effect more than a partial roasting. It is quite unsuited to the purpose under review, but is used in the chloridising roast of silver ores.

To suppose that what occupies 12 to 18 hours in a reverberatory can be accomplished instantaneously, as it were, in a shaft furnace is hardly reasonable. Nevertheless, experiments prove that a period of one minute suffices for the sweet roasting of a single pyrites grain under the most favourable conditions of temperature and access of hot fresh air. The problem

is to apply those conditions on an industrial scale to enormous numbers of grains simultaneously.

Another feature, which is of great importance economically, is the heat generated by the combustion of the ore. In theory, while the combustion of coal produces a temperature of about 5000° F., the combustion of iron pyrites should produce at least 1800° F., which is a higher temperature than is necessary for the operation of calcining, and allows a liberal margin for waste and imperfect combustion. This is verified in practice every day by the pile-roasting of cupriferous pyrites, where fuel is used only to ignite the mass ; and by the pyrites kilns of the sulphuric acid industry, where the ore, broken to egg-size, is burned in thick beds without any fuel whatever : the oxygen of the air admitted sufficing in both instances to support combustion once commenced. The essential differences in the conditions of the two cases of the acid works and the gold mill are :—(*a*) that complete oxidation is not attained in the former, but is generally essential in the latter ; (*b*) that the ore is in lumps in the one, and in powder in the other ; (*c*) that the sulphurous and other gases are utilised in the first, and wasted in the second.

But considerable quantities of dust pyrites are made in breaking and hauling the ore at acid works, and this fact led, many years ago, to the invention of a furnace for burning them, known as the Gerstenhöfer, which carries out to the full the principle of making the pyrites support its own combustion. The dust ore is caused to fall down a vertical shaft, its descent being delayed by a number of triangular crossbars ; these break up the stream of falling ore, and help to cause an exposure of every particle to the rising current of air admitted at the base of the shaft. When the furnace has once been heated to dull redness, no further fuel is needed to maintain it in operation on a richly sulphurous ore, such as ordinary pyrites. The sulphuric anhydride generated in the process is utilised for making sulphuric acid ; in fact, that is the main object of the furnace, though it has also found an important sphere of usefulness in dealing with cinnabar "fines." With the provision of more air inlets fitted with dampers, and the exercise of care

in controlling the temperature and feed, this long-known and well-tried furnace would meet the wants of many gold-fields where highly sulphuretted concentrates are produced ; and if the waste heat were utilised to warm the fresh air supplies, and the latter were regulated by a fan or other contrivance under complete control, no better, more simple, enduring, inexpensive, economical, and efficient furnace could be desired. If care be taken that the pyrites is reduced to 100 mesh, and the shaft be built 40 ft. high, an absolute dead roast can almost be guaranteed, while the wear and tear, labour and fuel consumption are only nominal. But special provision must be made for saving the valuable dust carried over by the draught.

Several modifications of this system have been introduced or proposed. Thus Fauvel's furnace has a series of sloping shelves alternating from opposite sides, with provision for heating the fresh air, while the products of combustion from the fireplace are prevented from coming into contact with the ore. So far this furnace has not yet been put into actual use at any gold mine, therefore working results cannot be quoted, but it looks highly promising.

A furnace which Cosmo Newbery was experimenting on in Victoria some years ago, and which he thought promised good results, was made with an inclined floor of "tiles" (presumably fireclay slabs), arranged so as to allow air to enter between them. They are about 6 in. wide, and are placed at an angle of 33°, so that the ore flows down in a constant stream. In the roof and under the floor are pipes, by which air, heated by the combustion of the pyrites, may be delivered over each tile.

A somewhat similar furnace was erected at Charters Towers, Queensland, but admitting all the air supply at one point, which is a serious error.

At the South German mill, Maldon, one of the best establishments in Victoria, a McDougall * shaft furnace is used. It has 8 circular superposed beds 6 ft. diam., and roasts 20 to 22 t. a week. The fireplace is at the side, and the fuel is wood. During a visit in 1899, the author was assured by the

* J. Mactear, Trans. Inst. M. & M., vi. 45 (1898).

manager of these works that the furnace did excellent duty at very low cost for fuel and maintenance, rendering the product fit for chlorination. But it would seem that the sulphurets here dealt with are particularly docile.

The pyritic ore of Mount Morgan consists of a gangue of quartz and silicified country rock, containing on an average 15 % pyritic mineral. Although the percentage of sulphur is therefore low, it is not easy to obtain the perfect roast required for a high extraction of the gold with a low consumption of chlorine. Various mechanical roasters were tried without success, until manager G. A. Richard invented his shaft or shelf furnace. This is said * to be "very economical in fuel, has a large output, and furnishes a thoroughly oxidised product. The ore moves automatically over inclined planes in such a way that the quantity of ore per square foot lying on the hearth increases as the end of the roast is approached, while the speed of travel decreases. The movement is assisted toward the end by jets of air, which serve also to supply oxygen. This is an expensive furnace to build, but it secures the low initial heat and final high temperature required by theory in a very perfect manner, and it makes hardly any flue-dust. The gases issuing from the furnace are exceptionally rich in $SO_2$ and low in oxygen, thus pointing to a high chemical efficiency."

According to J. D. Smith,† the furnace is 30 ft. by 12 ft., with 11 superposed hearths, the ore being moved down from shelf to shelf by intermittent air blasts through ½-in. tubes in the sides. Dust is arrested automatically, and flows back into the furnace. With wood at 15*s.* 2*d.* per ton, roasting to ·09 % sulphur costs 8*s.* 11*d.* per ton, as against 17*s.* 10*d.* in reverberatories, and 20*s.* 6*d.* in a Ropp.

(2) Rotating cylinders are known by a variety of names, especially in the United States. They may be divided into two groups—those with open ends, which are fed and discharged continuously ; and those with closed ends, which are fed and discharged intermittently.

* E. Hall, En. & Min. Jl., Oct. 7, 1899.

† New Zealand Min. Jl., Nov. 1898.

The original form of the continuously-operating cylinder is the Oxland and Hockin calciner, invented many years ago for treating arsenical ores in Cornwall ; thence it was introduced into America, and is there called the " White," or " Howell-White " roaster. It is shown in Fig. 105, and consists essentially of a wrought-iron cylinder, 20 to 30 ft. long and 3 to 4 ft. inside diam. (often an old boiler shell), lined with firebrick, and mounted in an inclined position, resting on wheels, which cause it to revolve slowly. The flame and gases from the fire pass through the cylinder in contact with the descending ore, which is fed in at the upper end by a screw conveyor. To increase the agitation and exposure of the ore, 4 straight parallel ribs of brickwork are carried down the cylinder for the greater portion of its length, and these alternately pick up the ore and drop it through the flame.

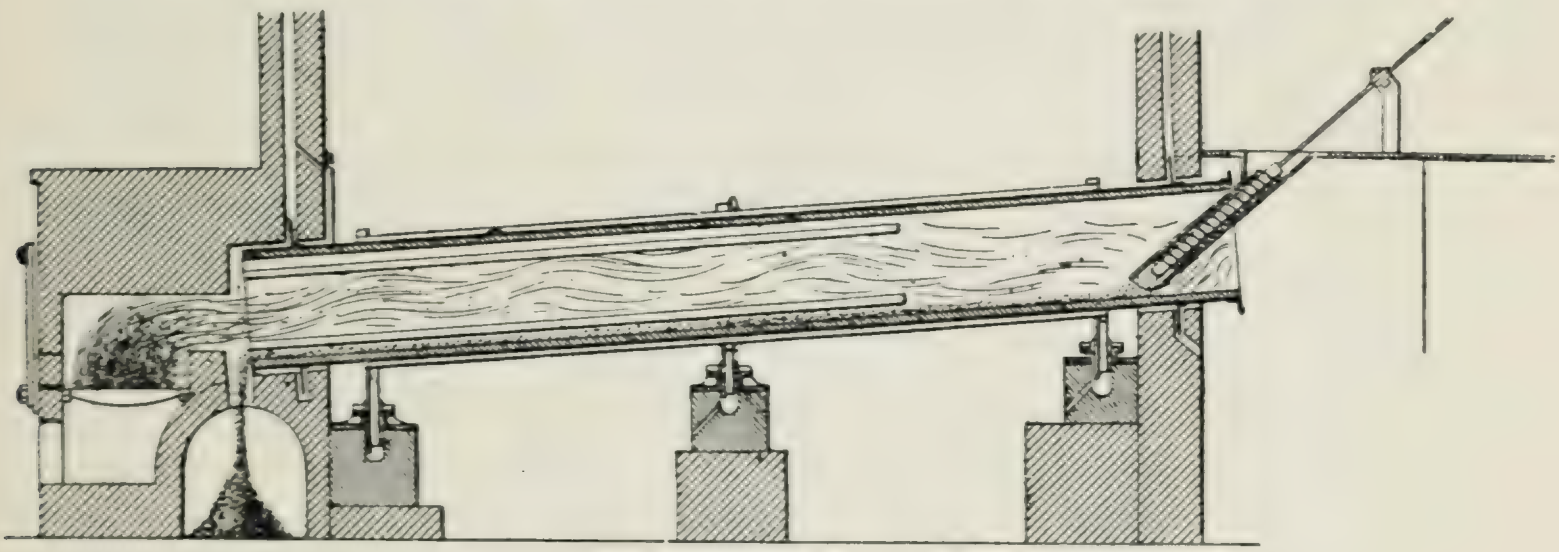


FIG. 105.—OXLAND AND HOCKIN FURNACE.

As made in the United States, this furnace is usually lined with firebrick only at the larger end, and cast-iron shelves are arranged spirally in it. Three standard sizes are recognised :—

| Dimensions. | Capacity. | Common Bricks. | Firebricks. |
|---|---|---|---|
| ft. in. | t. per 24 hours. | | |
| 24 × 40 | 15 to 20 | 26,000 | 1900 |
| 27 × 52 | 20 to 30 | 26,000 | 1900 |
| 27 × 60 | 30 to 45 | 28,000 | 2700 |

Sometimes an auxiliary fire is added for roasting the dust which escapes from the main furnace.

Working on arsenical pyrites in Cornwall, such a furnace roasts 20 to 25 t. per 24 hours, consuming only 10 to 20 lb. of coal per ton, and costing for labour but 1*s.* a ton, at Cornish rates of wages, but requiring 2 to 3 h.p. motive power.

With proper control of feed, fire and draught, it does excellent work, especially on ores which are not highly pyritic, or in which dehydration is the principal aim.

A similar furnace, 60 ft. long and 7 ft. diam., is used in Wales for calcining "white metal" down to 1 % sulphur, at a cost of less than 1*s.* 5*d.* per ton, including interest and repairs.

Mount Morgan, Queensland, is probably the most important example in the world of direct chlorination, over 2,000,000 oz. fine gold having been won in that way, and although the surface ores are thoroughly oxidised, it is found necessary to subject them to a preliminary calcination, which is conducted supposedly in order to dehydrate the limonite and kaolin, and so liberate the gold, although the heat used is much too low to dehydrate kaolin. The discovery during recent years that all the ore will give reactions for tellurium, probably explains more accurately why calcination is necessary. The oxidised ore is roasted in very large revolving cylinders with continuous discharge, resembling the Oxland and Hockin. They measure 57 ft. 6 in. × 7 ft. 6 in., exclusive of firebrick lining. The ore occupies 20 min. in passing through, and in this dehydration roast, 268 lb. wood, costing 2*s.* 2*d.*, are consumed in treating 1 t. ore.

Enormous quantities of auriferous mispickel have been calcined in this class of furnace at Marmora, Canada, preparatory to chlorination, therefore needing a sweet roast. Some modifications, however, are here introduced, which add materially to the efficiency of the furnace. Thus, the brick ribs are made in spiral form, and only occupy the upper 4 ft. or so of the cylinder, being replaced below by tile diaphragms, 4 in. wide, meeting in the centre, and dividing the cylinder lengthwise into 4 separate compartments. About half-way down, 1-in.

gaps are left in the diaphragms to enable the ore to stream through and fall across the air currents, which are supplied by a fan. These improvements are due to J. E. Rothwell, who claims, that with two furnaces thus equipped, one 30 ft. by 5½ ft., delivering into a second 60 ft. by 6½ ft., 48 t. mispickel are roasted per 24 hours, with 2 men per 12-hour shift, and only half the previous consumption of coal. Recently, cyaniding has replaced chlorination to a great extent on this field.

Working on ore carrying about 1 % each of sulphur and arsenic, crushed to 8 mesh, with labour at 10*s.* a day, and fuel at 14*s.* a cord (or 2*s.* 6*d.* a ton of ore), sweet roasting is done for 3 to 4*s.* a ton with this furnace in the United States, notably on the "oxidised" ores of the Potsdam formation in the Black Hills.

Other modifications which have been proposed in this widely-used furnace, with the object of increasing its capacity or efficiency, relate particularly to the feed. Thus, Pardee constructs a feed pipe with "diaphragm," as shown in Fig. 106, in which a boiler-plate flange *a*, in shape a segment of a circle, is so fastened to the inclined feed pipe *c* as to come within ½ in. of the circumference of the rotating cylinder *b*; by this means, the cold ore entering by the feed pipe is not exposed to the escaping current of heated vapours, which are deflected by the flange towards the top of the outlet. The proportion of flue-dust made is thereby greatly reduced, causing an economy of fuel and labour, and an increase in output. This was successfully worked for some years in Montana.

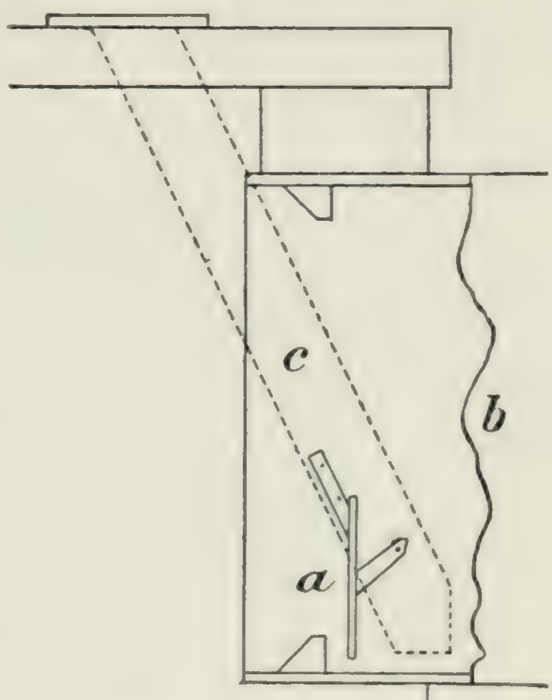



FIG. 106.—PARDEE DIAPHRAGM.

Rumsey's "diaphragm," also adopted in Montana, is shown in Fig. 107: A, screw conveyor; B, feed pipe; C, rotating cylinder. The upper end of the cylinder is made with a considerable flange projecting inwards, so as to form an annular

chamber; the pipe delivers the feed into this annular space, and here the ore remains for some time before it encounters the heat and draught of the furnace proper. The objects sought to be attained are the same as in Pardee's invention. Neither seems to have extended much beyond the place of its introduction, though beneficial results are claimed for both.

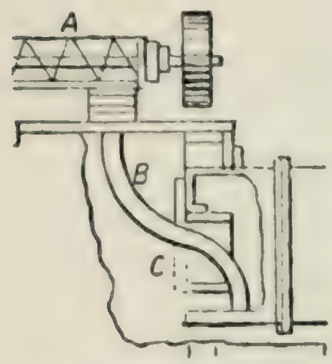



FIG. 107.—RUMSEY DIAPHRAGM.

The Molesworth furnace is made of iron, 15 ft. long, of conical shape, set on an incline with the smaller end downwards, revolving on friction pulleys. The products of combustion from the firebox under the smaller end circulate through flues arranged round the furnace, and do not enter the roasting cylinder proper, thus avoiding waste of oxygen. The ore is fed in at the upper end, and, during rotation, the particles are raised on shelves and dropped through the heated atmosphere. The energy of the oxidation is stimulated by the use of an oxygen-carrier in the shape of gaseous nitric oxide, generated by the action of sulphuric acid on nitrate of soda (a feature borrowed from the sulphuric acid industry); thus nascent oxygen encounters the ore at the moment when its heat is greatest, and when most of the sulphur and arsenic have been expelled, with the object of securing very perfect oxidation. With so much iron exposed to the action of the sulphurous gases, the wear and tear of this furnace may prove excessive.

The type of the intermittent revolving calciner is the so-called Brückner, which, according to J. Mactear, is simply the black-ash furnace of the alkali maker, invented in 1848 by Elliott and Russell, of Newcastle, improved by J. C. Stephenson, of the Jarrow Chemical Company, and further by Mr. Mactear himself, who converted the cylinder into a barrel, and quadrupled its efficiency. It would seem that when Brückner introduced the furnace into Colorado, in 1887, he added a series of internal iron pipes, with the object of augmenting the disturbance of the ore during rotation. Of course these

interior fittings, called a "diaphragm," were rapidly destroyed, and soon came to be discarded, so that the "improved"

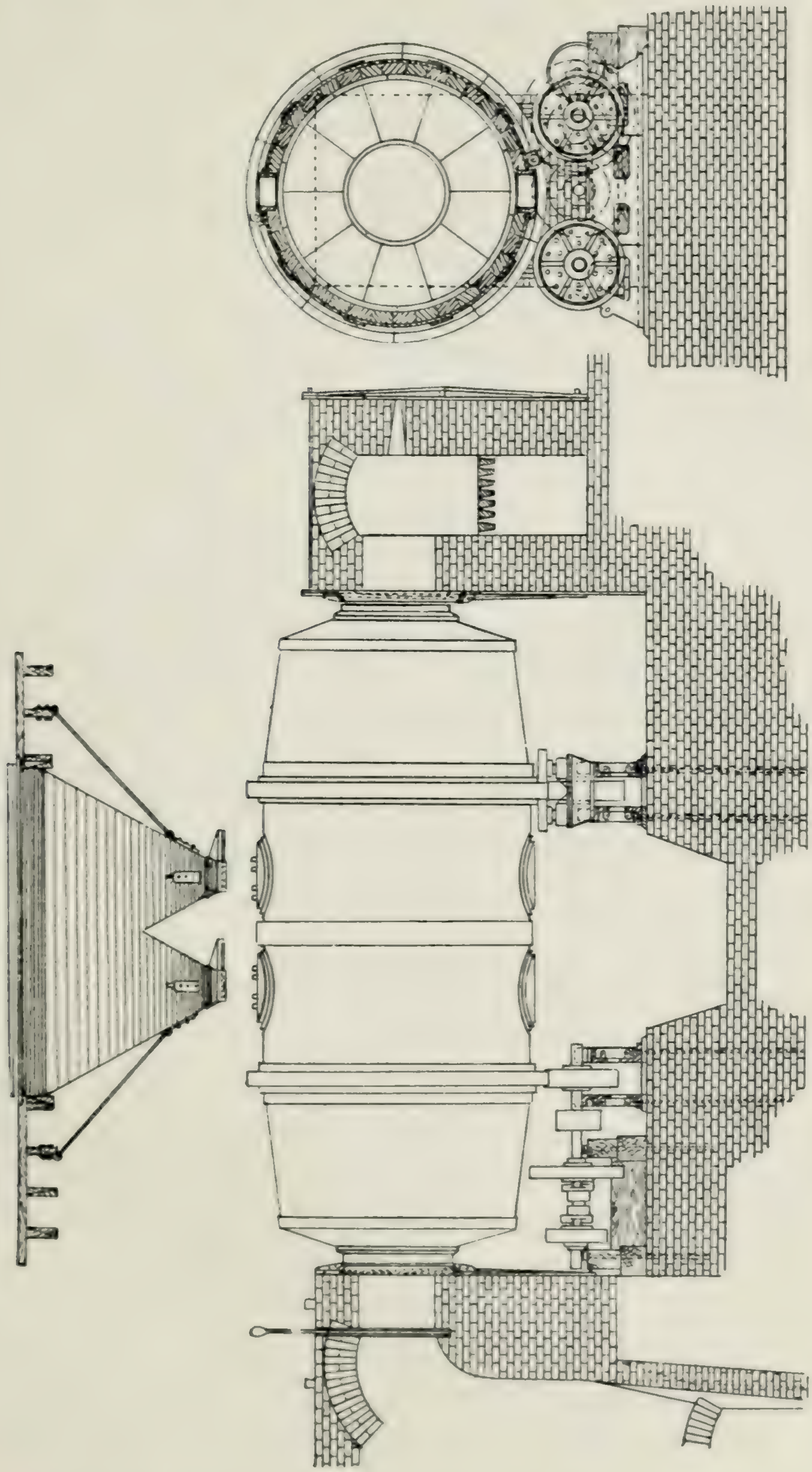


FIG. 108.—BRÜCKNER ROASTING BARREL.

Brückner of to-day is a reversion to Mactear's barrel. The rotation is effected by friction wheels (Fig. 108) in preference

to toothed gearing. Towards the ends, the cylinder is contracted by adding extra layers of brick lining, till the internal diameter is reduced from about 7 ft. to 2 ft. In modern practice, the length is preferably about 18 ft., and a charge about 7 t. The feed and discharge openings are opposite each other in the sides of the cylinder, instead of at the ends. One end admits the products of combustion from the fireplace, while the other discharges into flues leading to condensing chambers, and thence to the chimney. Obviously the ore can be retained in this furnace for any desired length of time, and that is the sole advantage it possesses over the Oxland. On the other hand, the charge is always unequally roasted, so that, to ensure all being sufficiently calcined, much must be over-roasted, incurring great waste of fuel. Moreover, the air-supply is insufficient; and the gradual increase of temperature, which is so desirable in an oxidising roast, can only be attained by alternately checking and urging the fire, added to which, the mechanical losses of dust are excessive. Quite as much motive power is required for this furnace as for an Oxland. The cost of calcination in the case of an ore carrying about 7 % each of sulphur and arsenic, crushed to 8 mesh, is not less than 6*s.* a ton, with labour at 10*s.* a day and wood fuel at 14*s.* a cord. These figures refer particularly to sweet roasting (for chlorination) the "blue" ores of the Potsdam beds in the Black Hills, which have lately been shown to contain tellurium.

It is very popular in America, and is there made with 4 receiving and discharging doors. The ironwork alone of a 7 × 18 ft. furnace weighs 15 t.; it needs 20,000 common brick for lining firebox and 10-ft. dust chamber, 1200 firebrick for lining furnace, 500 firebrick for lining firebox, and 3 barrels fireclay.

To mitigate some of the evils of the Brückner, Hofmann introduced the furnace shown in Fig. 109: A E, duplicate fireplaces—one at each end—so arranged that the firing can be alternated in either direction; B, rotating barrel; C, flues; D, dust chambers. Supposing the fire to be lit first in the right-hand fireplace, the draught passes through the barrel,

carrying the fumes down the flue into the dust-chambers beneath, and thence to the chimney. In due course, firing is done from the other end, the dampers being altered to create a draught in the opposite direction. An arrangement of dampers also permits the introduction of a supplementary current of cold air directly into the barrel, so that it will occupy a stratum between the ore and hot tainted vapours lying above it. This latter feature is claimed to greatly assist rapid oxidation—and no doubt it does assist—but hot fresh air would be better ; moreover, the same thing exactly has been done at one of the chlorination mills in the Black Hills (long since shut down), by placing a connecting cylinder between the firebox and the barrel, and cutting a hole in it,

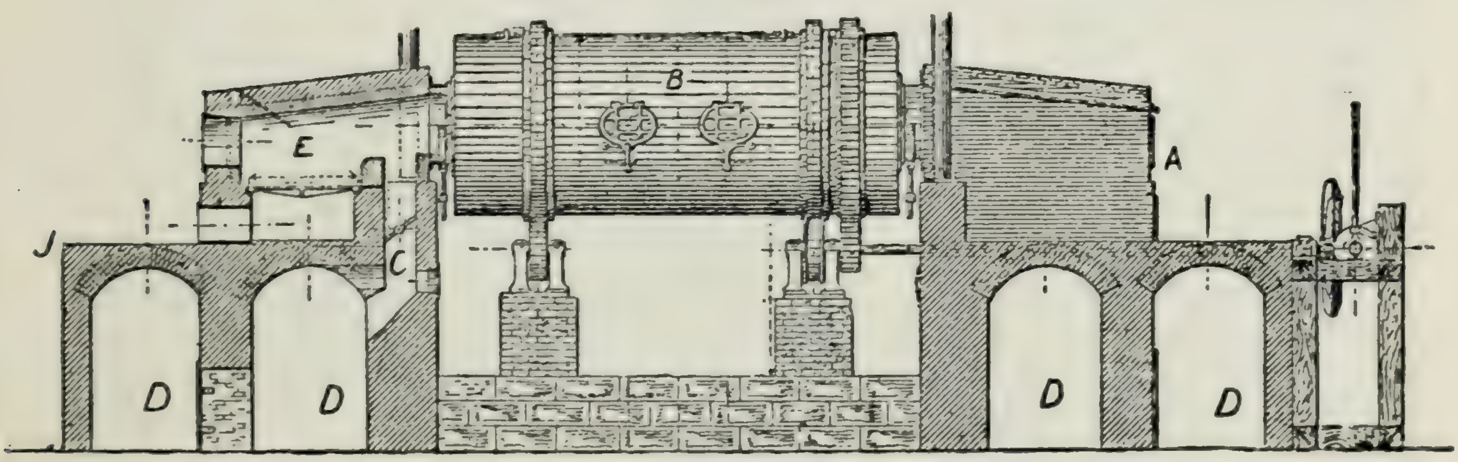



FIG. 109.—HOFMANN DOUBLE-FIRED BARREL.

with a sliding door, so that this part of the invention is not original. As to the use of alternate fires, the extra labour and waste of fuel probably entirely nullify the advantages of a somewhat more complete but still imperfect roast.

A novel form of rotating cylinder is the Argall roaster, consisting of 4 firebrick-lined steel tubes, 30 ft. long, nested together inside two steel tyres, as shown in Fig. 110. These tyres rest on and are driven by steel-faced carrying rolls suitably actuated. Their bearings are of the ball and socket type, lined with phosphor bronze, and they are supported by cast-iron sole plates resting on a heavy **I**-beam framework. The roaster is set on a slight incline, varied according to the duration of roasting necessary, and is held in place by vertical guide rolls. The firebox is a firebrick-lined steel shell, sup-

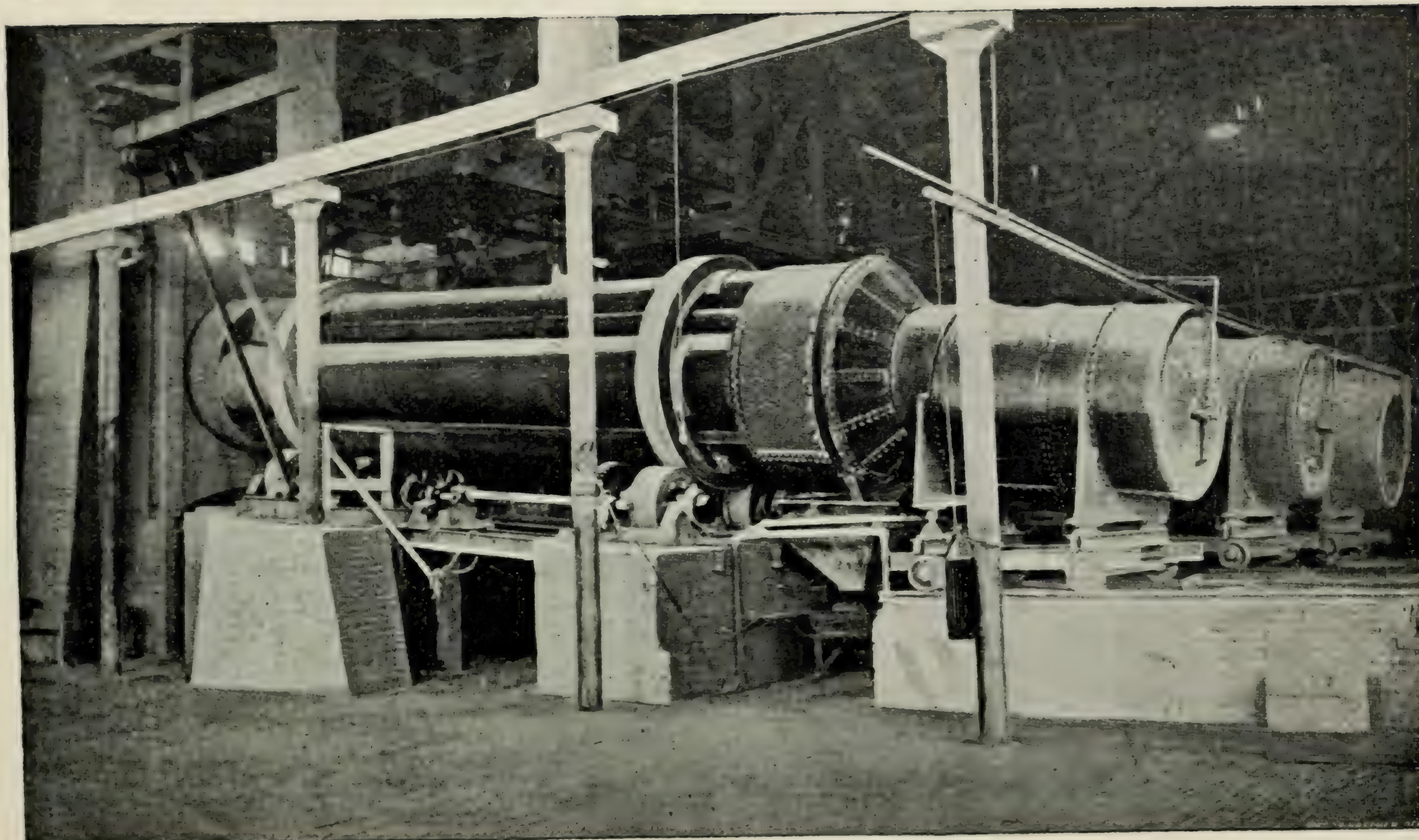


FIG. 110.—ARGALL ROASTER.

ported on a truck, so as to be easily moved out of the way. It will use coal, gas, or oil, the last being best in every respect. The ore is divided into 4 or more thin streams, and is brought into close contact with the oxidising flame in comparatively small tubes, resulting, it is claimed, in a very complete roast with great economy of fuel. The multitubular form of construction gives such strength that even in a furnace 60 ft. long no intermediate supports are necessary. Flue-dust is reduced to a minimum and a high terminal heat is secured. A furnace treating 45 to 50 t. per 24 hours of Cripple Creek ore, reducing sulphur to ·1 %, requires 1 h.p., occupies 756 sq. ft. (17 sq. ft. per ton per 24 hours), contains 63,600 lb. ironwork, costing 1200*l.*, and uses 14 t. firetile lining, 5000 firebrick, and 20,000 common brick. These roasters are made by F. M. Davis, Denver. They are used by the Economic Reduction Co., Eclipse, Colorado, and by the Boulder Perseverance Co., Kalgurli, W. Australia, both on telluride ores, and are endorsed by the Cyanide Plant Supply Co.

(3) Mechanical rabblers embrace a number of furnaces with stationary beds and automatic machinery for stirring the ore and exposing continually fresh surfaces.

Probably the oldest form is O'Hara's, consisting of two hearths with very low roofs, one placed above the other, and provided with endless chains carrying a series of "ploughs," arranged so that each set turns the ore in an opposite direction, from side to centre and *vice versâ*. The ore is made thus to gradually approach, first the orifice leading from the upper to the lower hearth, and then the fire-end of the latter, where it falls into a pit. In efficiency, capacity, cost of operating, and wear and tear, this furnace is quite out of date.

In the Spence furnace,* the travelling ploughs are replaced by rakes having a longitudinal reciprocating motion, actuated by machinery. In other respects it resembles the O'Hara. Except that the number of superposed hearths may be more than two, the objections on the score of efficiency, capacity, cost of operating, and wear and tear are the same. Neither of these furnaces, in fact, is fitted to perform sweet-roasting,

* Described in great detail in Peters's "Copper."

and in several instances where they have been built with that object they have been pulled down after trial.

The Brown-O'Hara, which soon developed into the Brown-Allen and thence into the Brown "Horseshoe" (Fig. 111), dispenses with the superposed hearth and adopts a single

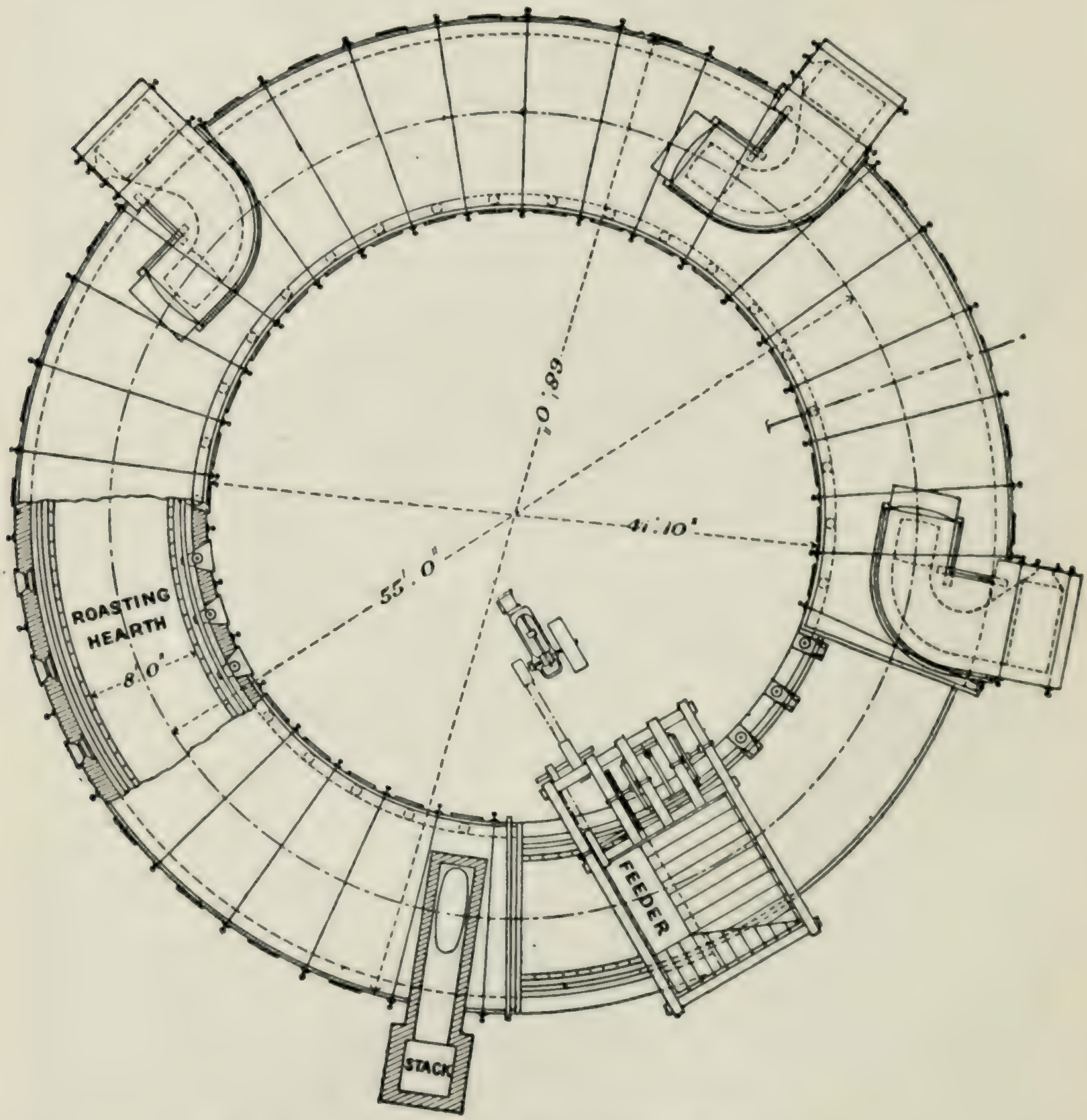



Fig. 111.—Brown's Horseshoe Furnace.

hearth built in annular form, with a section equal to about one-fifth of the total circumference removed, whence the term "horseshoe" has been derived. This form has some distinct advantages. The shape gives the greatest capacity in the smallest space, and permits of centrally-driven rotating rabbling gear, while the parts exposed to the fire and gases

are beyond these injurious influences during one-fifth of each rotation, thus greatly lengthening their lives. In cross section, the main walls and arch are exactly the same as in a common reverberatory furnace. Protection from furnace heat is obtained for the mechanism used in stirring the ore by partitions or walls built from the hearth and extending nearly to the centre between the hearth and the roof of the furnace, forming annular chambers. Where a high heat is required, continuous walls of fireclay tiling are also built into the roof, directly above and projecting downward toward the lower walls, leaving a space to allow stirrer-arms to pass freely between lower and upper walls. Thus the tracks and moving mechanism are so thoroughly protected that they are not affected by the heat and fumes of the furnace. Carriages for conveying the stirrers move upon open tracks, and are operated by a continuously-running cable, supported on grooved wheels, and held taut by a tightener weight and pulley. The cable-carrying wheels have their entire annular surfaces all the time revolving in the open air. The carriages supporting the stirrers have an L-shaped frame with double-flanged bearing wheels travelling on a T-rail in the inner annular chamber, and a broad flat-faced bearing wheel rolling on a smooth track-way, usually made with T-iron, in the outer annular chamber. A gripping device is attached to each carriage. This consists of a stationary and a movable jaw, the latter operated by eccentric shaft and lever, so as to be closed upon the cable or released automatically by engagement with stops, which are brought into operation by the movement of the carriages around the furnace. In operation, half the carriages are traversing the furnace, and half are resting in the cooling space, at any time. The forward moving carriage comes into contact with the rear one at rest, and so pushes both carriages into the cooling space, until, by automatic grip and release, the forward resting carriage takes up the stirring, and the forward moving carriage comes to rest. With this provision for cooling, the stirrers do not get over-heated even when subject to a high furnace heat. While at rest, the carriages are disconnected from the furnace mechanism, and can be removed without at all interfering with the work of

the furnace, and an independent throw-out can be operated to stop the forward moving carriage before the forward resting carriage is engaged by the cable grip, and so all may be removed from the furnace at will.

The makers, Fraser & Chalmers, claim for this furnace that it is very simple in construction, requiring no more brick and no more ironwork for binding-rods and buckstays than the ordinary reverberatory furnace of the same length of hearth; costs to build 50 % less than any other mechanically stirred roasting furnace of the same capacity; the operating mechanism is easy to manage, and not liable to get out of order; the stirrer carriages, standing half the time in the open air, are kept thoroughly cooled, and are at all times perfectly accessible; 1 man on a shift can manage the machinery and fires for a furnace of 40 to 60 t. daily capacity; there is no dust raised by falling ore, and no loss of heat; the feed is automatic, introducing any required quantity of ore with the passage of each stirrer, the amount being governed by a counterpoised lever, which weighs each charge; all journals exposed to the heat are fitted with ball or roller bearings, requiring no lubrication.

In the rebuilt works of the Golden Reward Co., Black Hills, a Brown horseshoe furnace has replaced the Brückners formerly used. It has * a roasting hearth 180 by 8 ft., and a cooling hearth 78 ft. long. It deals daily with 65 t. of "blue" Potsdam ore, crushed to 30 mesh, and carrying 2½ to 8 % sulphur, and delivers it with ·3 %. The total costs, daily (24 hours) and per ton, are as follows:—

| — | Per diem. | Per ton. | % |
|---|---|---|---|
| | £ s. d. | s. d. | |
| Fuel, 6 cords wood at 13*s.* 8*d.* | 4 2 0 | 1 3·13 | 71·84 |
| Labour, 2 men at 10*s.* 4*d.* | 1 0 8 | 3·81 | 18·09 |
| Power, 5 h.p. | 0 5 2 | ·97 | 4·60 |
| Oil, light, repairs, etc. | 0 6 3 | 1·15 | 5·47 |
| | 5 14 1 | 1 9·06 | 100·00 |

* En. & Min. Jl., July 4, 1896.

These figures include roasting, cooling, and conveying to chlorinating barrels.

According to Manager H. C. Callahan,* of the Lake View Consols, the Brown straight-line furnace is in operation there, while the Associated mill uses a Ropp (Fig. 112), which has been very successful at the Port Pirie works of the Broken Hill Proprietary Co.

At the Colorado City works, Ropp furnaces 100 ft. by 14 ft.

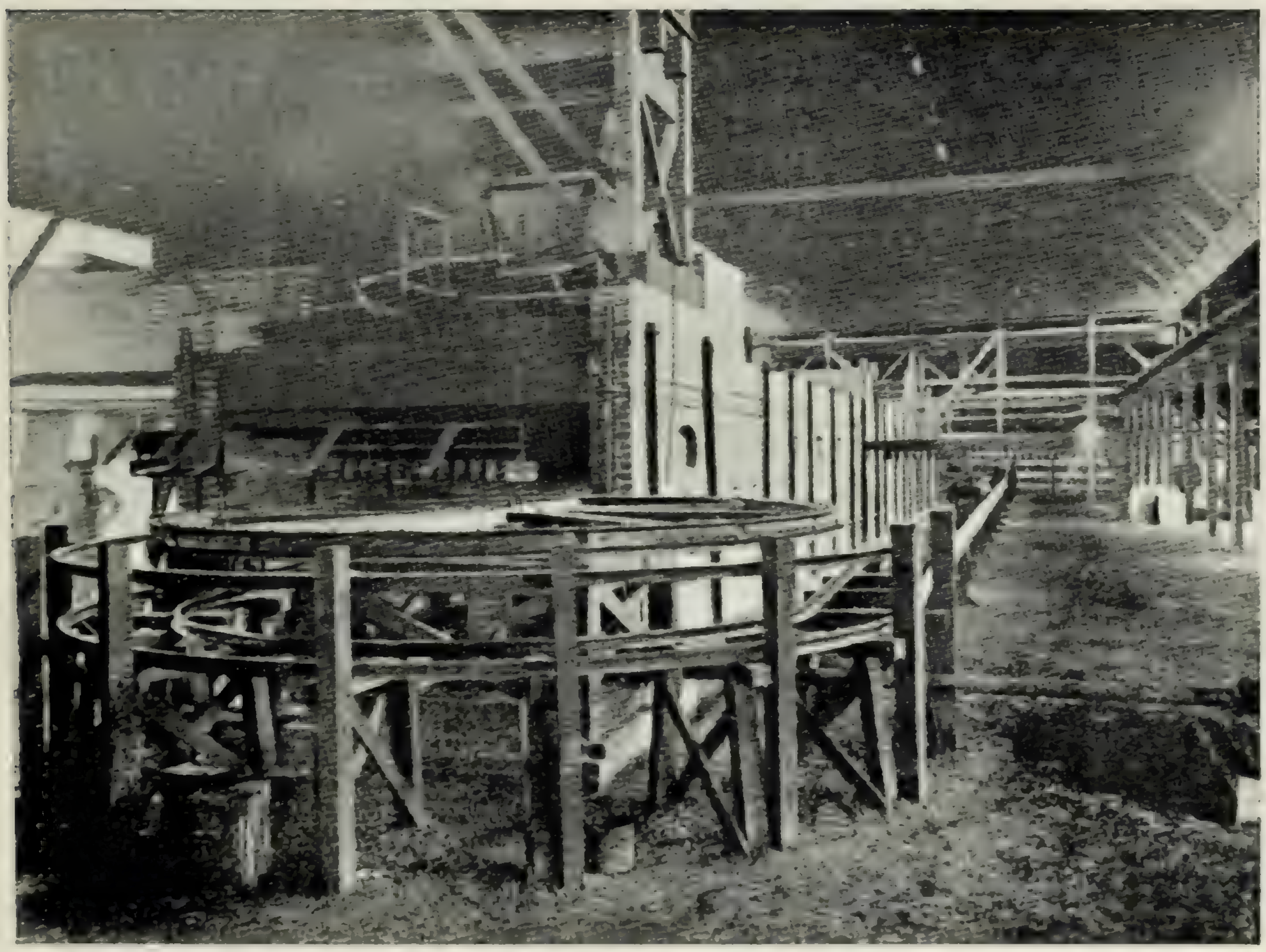


FIG. 112.—ROPP FURNACE.

deal daily with 90 to 94 t. each of 12-mesh ore carrying 2·6 to 3·5 % sulphur, reducing it to "·02 % sulphur after deducting soluble sulphates" (H. V. Croll); they use oil residuum as fuel. The building covering these furnaces is 156 ft. long and 90 ft. wide. The flues of the 3 roasting furnaces and that of the drier connect with a brick dust-chamber 10 by 12 by 150 ft. long, leading into an iron stack 9 ft. diam. and 150 ft. high. The pulp is delivered from the crushing department by belt

* Mining Journal, March 31, 1900.

conveyor to four iron storage bins over the roasting furnace. From the bins it is taken to the furnace by automatic feeders. The roasted ore is discharged from the furnace into steel ore cars, and trammed to a Durant automatic cooler, which is located in a pit outside of the furnace house. From the automatic cooler the roasted ore is conveyed to a bucket and belt elevator, which lifts it to a hopper above the charging floor of the chlorinating barrel room. From this hopper it is drawn into cars, weighed, and distributed to the charging hoppers over the barrels.

At the Great Boulder, good results are anticipated from an Edwards furnace, in which the ore is brought forward by water-jacketed rabblers.

The Perseverance is experimenting with a Wethey furnace, which resembles the Brown straight-line.

The Edwards furnace is much used in the older Australian colonies, having been first installed by the Edwards Pyrites Co. at Ballarat and Bendigo. It is a mechanical furnace, built of iron lined with brick, as shown in Fig. 113. It consumes less than one-third of the fuel that the hand-worked reverberatory furnace requires to roast the same amount of ore. The grade of the furnace may be altered to suit all classes of ore, or to prevent it being discharged if not properly roasted. The output varies with the percentage of sulphur or arsenic, but ordinarily ranges from 50 to 120 t. per week from each furnace.

The ore is automatically fed, and the roasted ore is similarly conveyed to the cooling shed, so that labour is reduced to a minimum. One man per shift is attending to 4 of these furnaces at the Tasmania Co.'s plant at Beaconsfield, Tasmania. He does all the firing and attends to all the requirements of each furnace.

This furnace was specially designed for roasting highly-concentrated refractory ores, as will be noticed by the motion of the rabbles, the slowest of which makes 1 rev. per min.; they travel in opposite directions, cutting each other's circles, thus pushing the roasting ore forward and backward. At the same time the ore takes a zig-zag course, crossing and re-

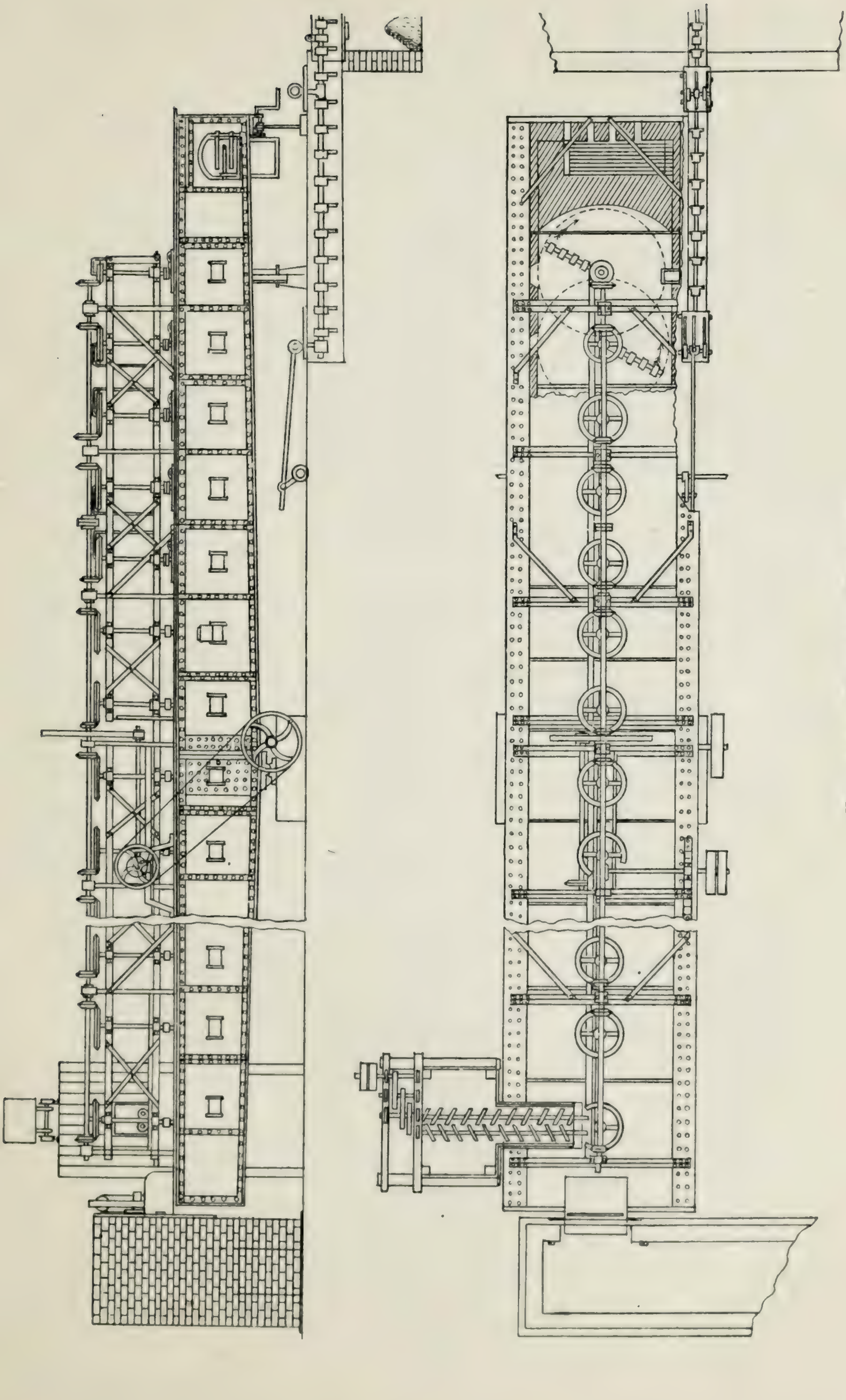


FIG. 113.—EDWARDS FURNACE.

crossing the furnace toward the discharge end, and it can be hastened or checked by either raising or lowering the discharge end by means of the screw, the pitch of which is two threads to the inch. The furnace is so balanced that the workman can run it up and down with the greatest celerity and freedom.

The condition of the interior and the progress of the roasting can be seen through openings on either side of the furnace. At Ballarat, the fuel is producer gas ; at Bendigo, wood. Coal may also be used. The discharge pipe works in a sleeve, to prevent escape of dust, and delivers to a pipe conveyor which transmits to a cooling floor. Cast-iron spindles carry the rabbles, which are hollow, and cooled by a current of water. The rabbles can be changed in a few minutes, and prove to be very durable. At the Tasmania mill, labour amounts to only 7*d.* per ton, while fuel costs appreciably less than with hand-rabbling.

At the Mercur mines,* Utah, the talcose oxidised ore is calcined to red heat to drive off water of hydration ; this destroys its plasticity and renders it "brickified." The furnaces used are Holthoff-Wethey, 100 ft. by 12 ft. ; each treats 145 t. of ¼-in. ore per 24 hours, consuming 8 t. of coal. The ore occupies 6 hours in passing through. The base or pyritic ores, containing 2 to 5 % sulphur and 1 to 2½ % arsenic, are roasted down to ·1 to ·15 % arsenic in Jackling straight-line furnaces, the process requiring 8 hours, and the capacity of each furnace being 70 t. of ⅛-in. ore per 24 hours, at a cost of 7 t. coal.

This furnace is shown in Fig. 114, and is said to possess the following novel features. The roasting hearth is independent of the main structure, so that buckstays, tie-rods, brackets, rails, etc., are not affected by expansion or contraction of hearth or arch. An automatic stop cuts off motive power in case of breakage of either propelling chain. Special elevators, deriving power from idlers on return end of furnace, discharge the roasted ore on the upper corrugated-iron floor, where it is cooled by water sprays. The fireboxes are near the discharge end, on each side ; the gases pass over the bridge walls into

* H. L. J. Warren, En. & Min. Jl., Dec. 30, 1899.

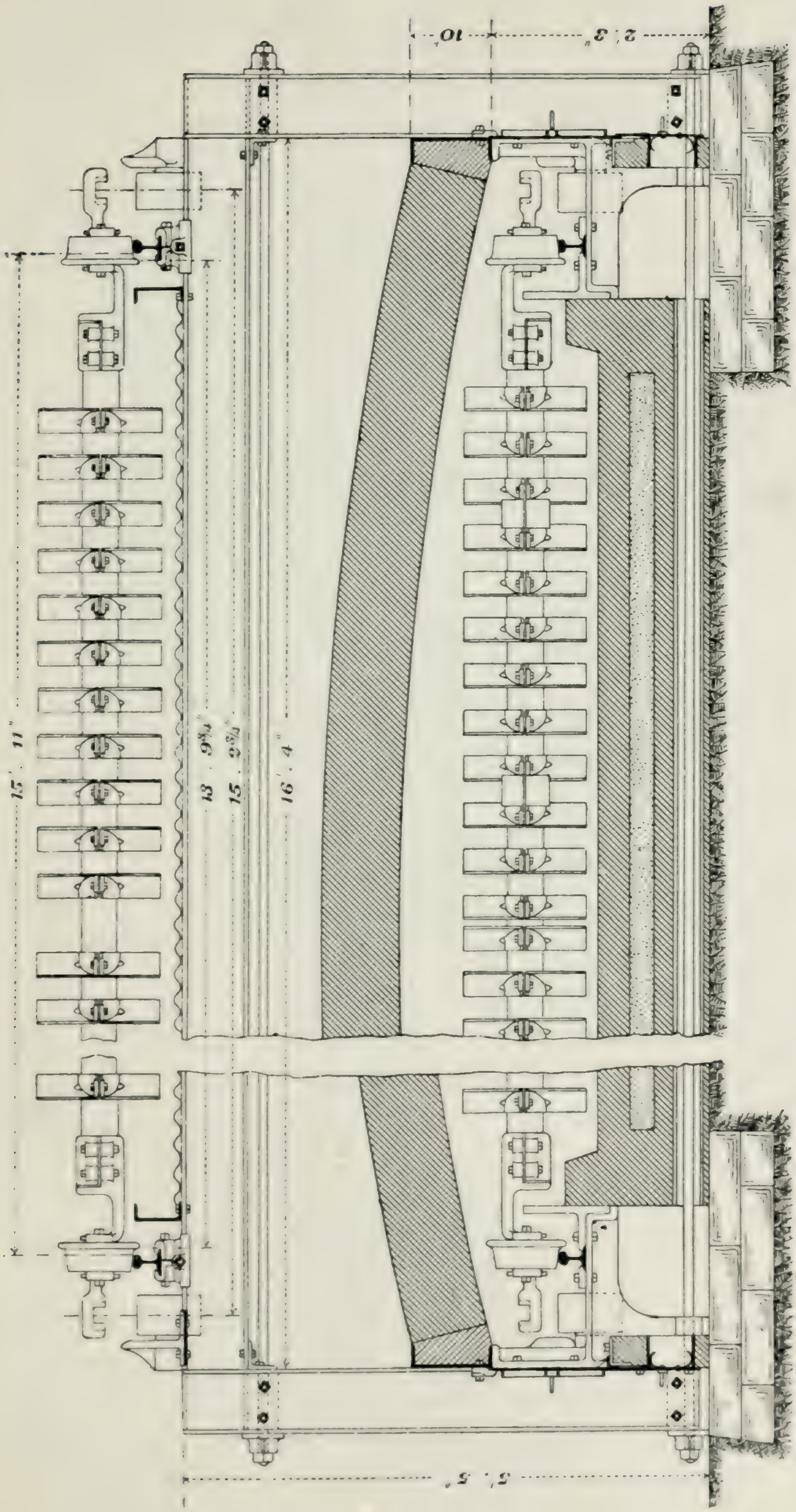



FIG. 114.—JACKLING FURNACE.

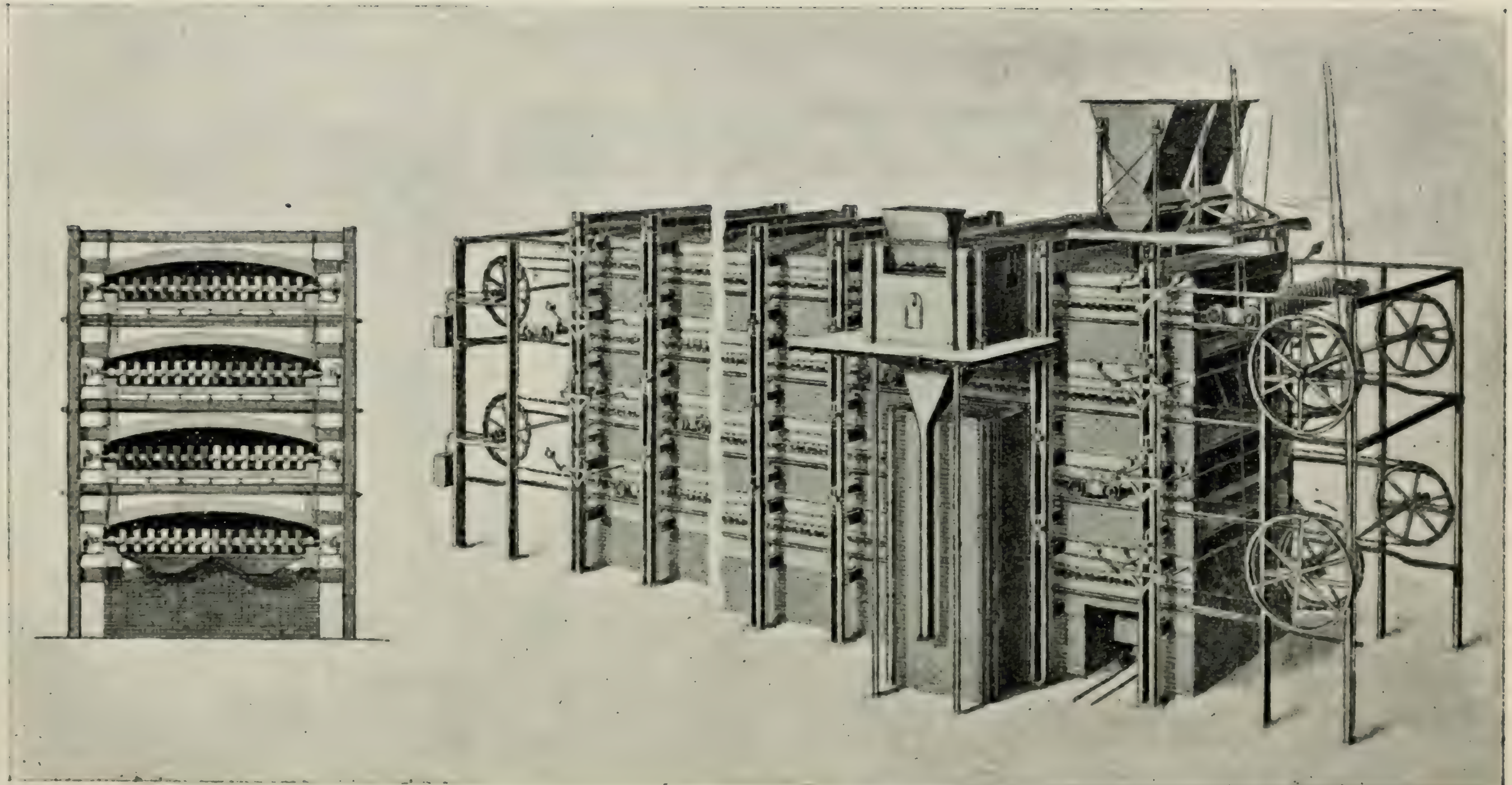


FIG. 115.—WETHEY 4-HEARTH FURNACE.

the main roasting chamber, and thence to the stack near the feed end. The hearth is 120 ft. by 12 ft.

Of multiple-deck furnaces the Wethey, Fig. 115, is a good type. Where room is limited, it can be constructed with 4 hearths, one over the other, each hearth being 50 ft. long by 10 ft. wide. If, however, there is ample floor space, it can be constructed with two hearths, each 100 ft. long by 10 ft. wide. Either of these furnaces has a hearth area of 2000 sq. ft. There is no limit to the dimensions of this type of furnace. The tracks, over which the carriages operating the stirring blades run, are situated on each side of the hearths, and are readily accessible. All the working parts and train work are on the exterior, and can be reached without the necessity of closing down the furnace. The fireboxes and flues can be connected at any point. For heavy sulphide ores, the fireboxes are preferably situated near the feed hoppers, so that the ore may be ignited as soon as it enters the hearth. The heat then travelling with the burning ore is sufficient to continue the process of desulphurisation without additional firing, and the ore travelling over the hearths, dropping from floor to floor, is finally carried out by the ploughs in a thoroughly calcined condition. A 4-deck furnace with 50-ft. hearths should calcine 60 t. daily from 40 % sulphur, down to 6 %. For ores carrying zinc, lead, and 18 to 28 % sulphur, more firing is necessary, as the sulphur present does not assist the process of desulphurisation ; then a long furnace with two floors is preferable, fireboxes being connected to each floor. If the ore becomes sticky and clings to the stirring blades, it can be easily removed when they come out of the hearth and pass on to the other floor.

The Holthoff-Wethey furnace, Fig. 116, is adapted to a class of ores for which the Wethey is not suited, and aims at a dead roast. The hearth is 12 ft. wide and 100 ft. long. The arch is held firmly between two heavy **I**-beams suspended from above from beams resting on channel-iron posts. The general construction is such that the expansion and contraction of both the brickwork and the ironwork are provided for. The furnace being of rectangular form, and supported by the

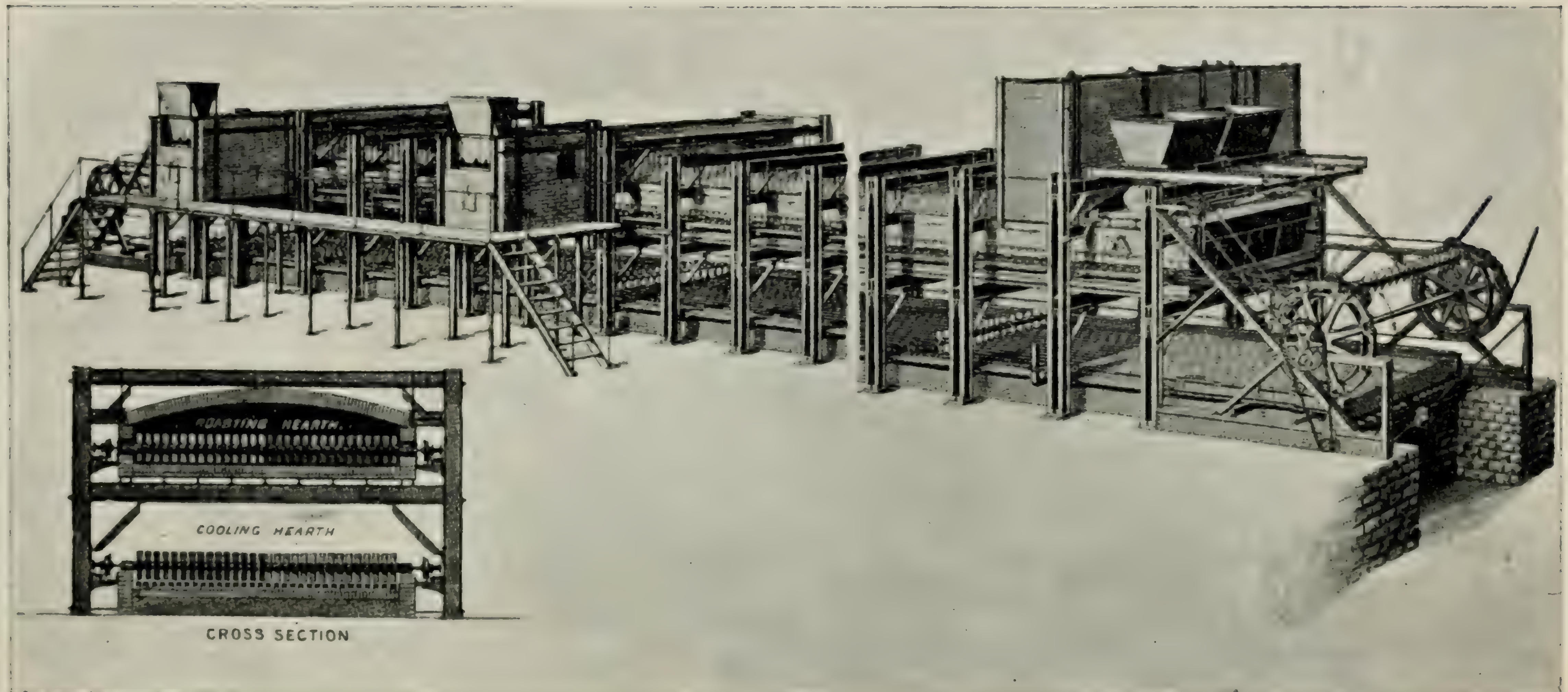



FIG. 116.—HOLTHOFF-WETHEY FURNACE.

same means on both sides, is less liable to injury from expansion and contraction than other types in which the arch is supported on one side by iron and on the other by brick. The design is such that no ironwork whatever is exposed to the fire except the rabbles. These are constructed of heavy pipe, to which steel ploughs are clamped, and are supported at both ends by carriages, and driven by wire rope and sprocket wheels. The carriage is so constructed that the rabble pipe and ploughs can be lifted out and replaced without disturbing the connection between the carriage and rope. Half of the ploughs on each rabble are set at one angle and half at an opposite angle, thus overcoming all end thrust. The fireboxes shown are arranged for burning slack coal, but can be modified to suit any fuel.

One of the principal new features in this furnace is the addition of a cooling hearth, for cooling the ore and delivering it to a conveyor, whence it may be transported to any part of the mill by machinery, thus not only saving cost of manual labour required to transport the ore from furnace to cooling floor and from cooling floor to the point where it is to be treated, but obviating the necessity of a large cooling floor and its accompanying building.

The ore is charged into the furnace at the driving end, and after travelling the full length of the roasting hearth is dropped to the cooling hearth and carried back to the charging end, thus allowing the ore the same length of time to cool that was required to roast it. To facilitate the cooling of the ore and ensure its temperature being sufficiently low to prevent injury to the conveyors and elevators, water pipes are laid the full length of this hearth flush with the top ; the amount of water necessary to effect the desired cooling is regulated by the discharge from these pipes. The pipes are laid loose in grooves between the brick in such a manner that they can expand and contract without injury to the hearth or themselves.

A valve shuts off the stream of ore from the feeder while the rabble is passing under it. Where the rabble passes through a stream of ore falling into the furnace, a certain amount of ore lodges on the rabble, and is carried through the

furnace and discharged in a raw state. This amount may not be great for each, but where a rabble passes through this stream once every minute or 1½ minute, it amounts in all to enough to increase the sulphur contents to quite an appreciable extent.

The space required for a given capacity is claimed to be much less than for any other type of furnace, and when the space ordinarily required for cooling floor is taken into consideration it is obvious that a great deal can be saved in the installation of a plant. The cost of a complete furnace per sq. ft. of hearth surface is also, it is believed, much less. The ironwork being made self-supporting, less brickwork is required. A 100 × 12-ft. furnace, as shown, will require 60,000 common brick and about 5000 firebrick. The furnace is made by the Edward P. Allis Co., of Milwaukee, Wis.

One of these furnaces at the Puget Sound Reduction Works, Everett, Washington, roasts 59 t. per diem, consuming ·198 cord wood per ton, and reducing the sulphur contents from 29·1 % to 4·4 %.

The roasting furnaces at the Golden Gate mill, Mercur, are fired by a mixture of producer gas and water gas, generated by a Loomis plant. This contains 4 generators and 2 coolers, one cooler being connected with two generators. One pair of generators makes water gas while the other pair is making producer gas, the alternations being carried out in the following manner :—Air is first drawn down through the fireplace, and the coal in the generator is brought to a state of incandescence ; the air is then shut off, the top doors are closed, and steam is admitted to the generator from below. Passing through the incandescent fuel, the steam is converted into water gas, which is drawn off at top, passed through a cooler, where it raises steam by giving off its heat, and collected in a main holder. After about 5 minutes the process is reversed by shutting off steam from the bottom and admitting it at the top, while air is drawn in and passed with the steam through the fuel. This forms a producer gas free from tar, which is passed through the cooler, and joins the water gas in the holder. Besides securing a uniformity of roasting quite

impossible with direct combustion, the saving amounts to 30 %.

The Pearce "turret" furnace, shown in Figs. 117 and 118, claims attention as being the invention of a prominent me-

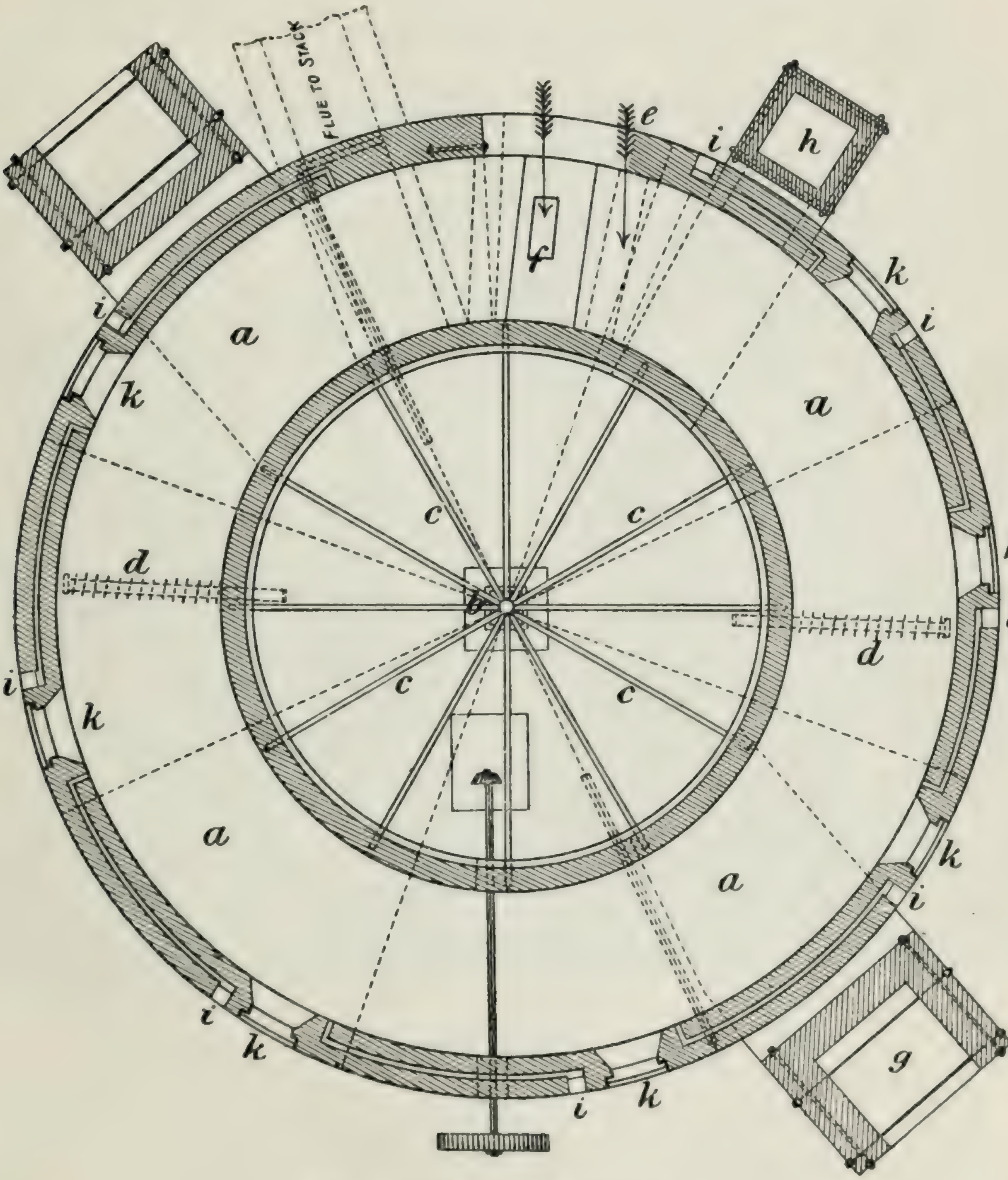



FIG. 117.—PEARCE FURNACE.

tallurgist, managing one of the largest smelting works in the United States, at Argo, near Denver, where it has been adopted to the exclusion of all others. It consists of an ordinary reverberatory hearth *a* built in circular form, the centre of the

circle being occupied by an iron column *b* supporting radiating arms *c* which carry rabble-blades *d*. The ore is fed in at *e* mechanically, and after traversing the whole circle of the hearth at any desired speed, falls by gravity into a pit *f*. Air is forced through the pipe arms and discharged against the rabble-blades, performing the double duty of cooling the iron-

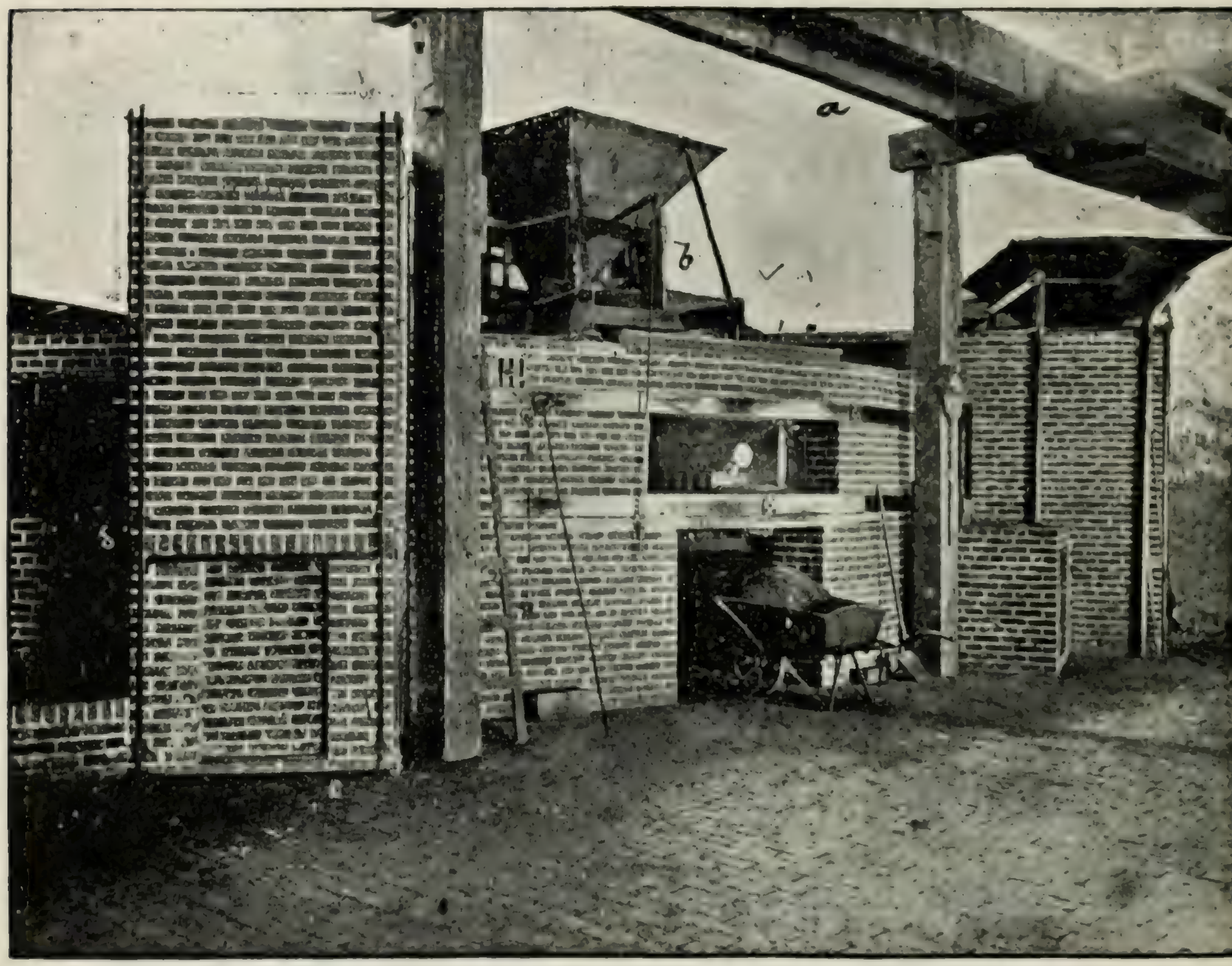



FIG. 118.—PEARCE FURNACE.

work and furnishing heated air to the roast. Two or more stepgrate fireplaces *g*, automatically fed, supply heat. The space beneath the hearth is utilised as a dust-flue *h*. The cost of a 36-ft. furnace is 800*l*. for ironwork and erection, based on Colorado prices, with 300*l*. for royalty. Repairs are confined to renewal of rabble-blades every 3 or 4 weeks. One man per shift can feed fuel and ore, remove roasted ore, and attend to machinery of one furnace, if assisted by the mechanical

arrangements shown in Fig. 118, which consist of overhead tramway *a* for bringing ore and fuel, and feed hopper *b*. The air supply can be controlled not only through the perforated arms, but also by additional inlets *i*, *k*. The roasting can be adjusted to any desired degree, from a dead roast to any percentage of sulphur, and to fusion or sintering. For chloridising, salt can be added at any point; and the furnace can be used for drying or for cooling. The dust is much reduced. Its capacity varies from 10 to 60 t. per 24 hours. Very inferior fuel can be used if forced draught is applied. No other furnace is so well designed for keeping the roasting in perfect control, and in that direction as well as in economy lies its superiority. The cost per ton ranges from 2*s.* to 3*s.* 6*d.* At the Delano mill, Colorado, operating on a telluric ore crushed to 20 mesh and carrying 2·2 % sulphur, and reducing it to ·04 to ·4 %, a Pearce furnace 40 ft. by 8 ft., using inferior lignite for fuel, is capable of dealing with just 50 t. a day; and at the Colorado City works, a similar furnace treats 55 to 60 t. a day of 12-mesh ore, reducing its sulphur from 2·6 to 3·5 % down to ·02 % "after deducting soluble sulphates" (H. V. Croll), using oil residuum as fuel with most satisfactory results.

At Delano, it has been found advantageous to cool the rabbles by water instead of air, and forged steel teeth have replaced the usual blades for stirring, thus securing much longer wear without the drawback of admitting a volume of cold air to lower the temperature of the furnace. The teeth are made plough-shaped.

A modification of the automatic rabbling furnace, which introduces the advantages of the drop action, is Denny's, shown in Fig. 119: *a*, arm revolving with shaft *b*, and carrying loose rake *c*; *e*, base plate; *g*, firebars; *h*, firebridge; *k*, fireplace; *l*, hopper; *m*, bevel wheels. A self-acting grate at the bottom of the hopper admits a charge at intervals. On the top shelf, drying only is effected. The next shelf retains a 4-cwt. charge for 20 minutes or so, during which time the ore is continually stirred by the loose rake hanging from the revolving arm. A slide operated by a boy opens a slot in each

iron shelf, which allows the charge to pass downward at intervals till it finally reaches the lowest shelf. Besides the air entering by way of the fireplace, additional supplies are furnished at about one-third of the height of the furnace, after being heated in a capacious air-space immediately over the fireplace. While possessing some good features, notably in the drop action and in lessening mechanical loss of dust by the numerous impediments to the draught, the destruction of the internal ironwork must be exceedingly rapid, making the

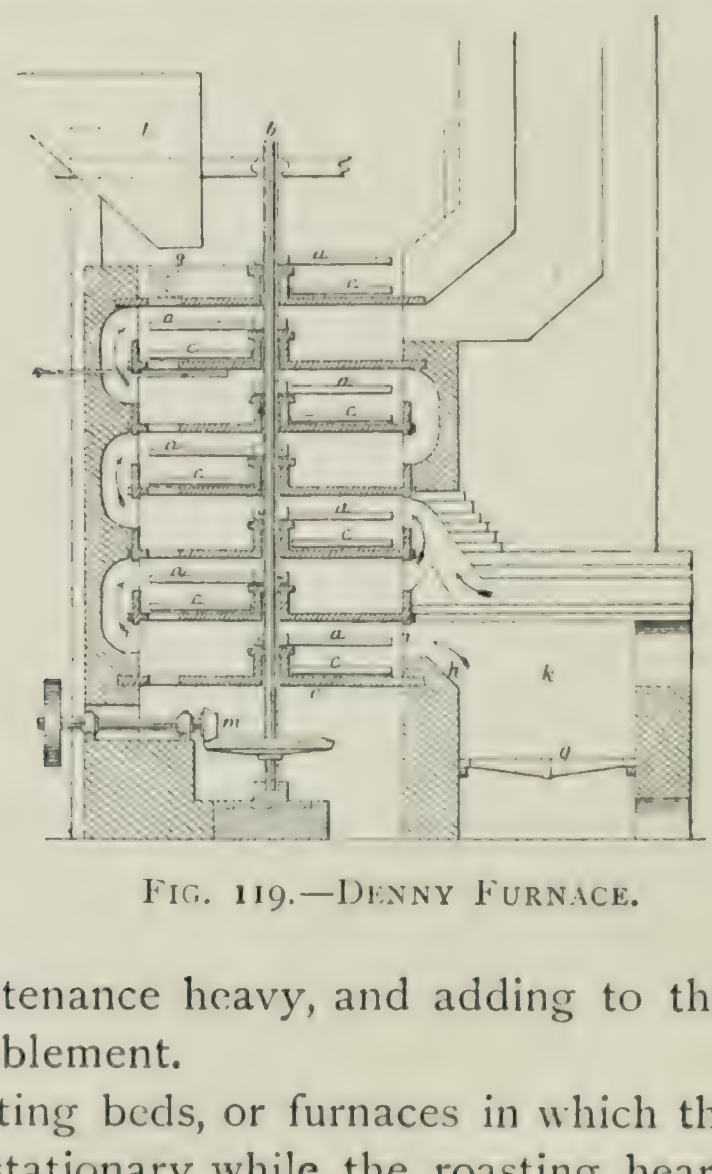


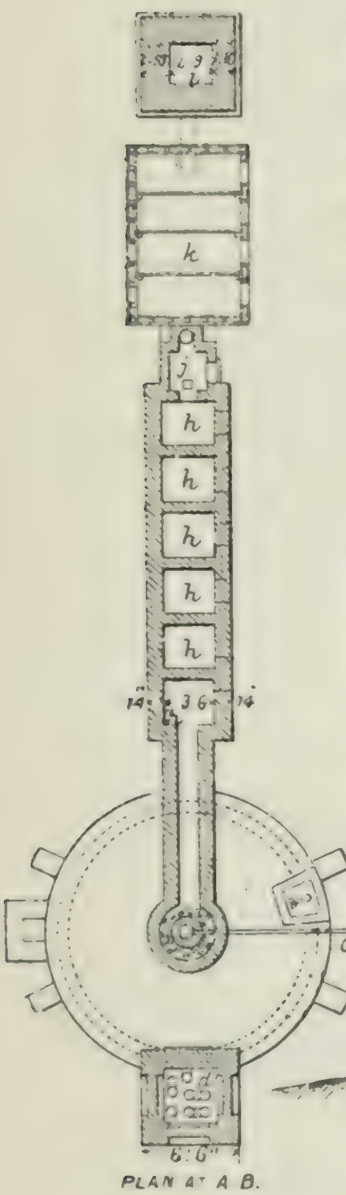
FIG. 119.—DENNY FURNACE.

cost of maintenance heavy, and adding to the risks of stoppage by disablement.

(4) Rotating beds, or furnaces in which the rabbling mechanism is stationary while the roasting hearth revolves, are also typified by an old Cornish calciner, known as the Brunton. This is essentially a circular reverberatory, with a firebrick hearth laid on a slightly convex iron table, which is made to revolve slowly. The ore, fed in at the centre of the low domed roof, is gradually worked towards the outer edge of the hearth by numerous iron "flukes" or blades projecting downwards from the roof. In treating arsenical pyrites in Cornwall, this furnace has an average capacity of 4 to 5 t. per 24 hours,

employing one man per 12-hour shift, and consuming $1\frac{1}{2}$ to 2 cwt. coal per ton calcined, in addition to that consumed in generating power to give motion to the hearth. Therefore it cannot be regarded as economical.

Nevertheless, a few modern furnaces have been founded on this type. Serjeant and Flude's, shown in Fig. 120, was introduced some years since in Victoria, and has the laudable object of utilising the sulphurous gases for making sulphuric acid and recovering the arsenic as arsenious acid (white



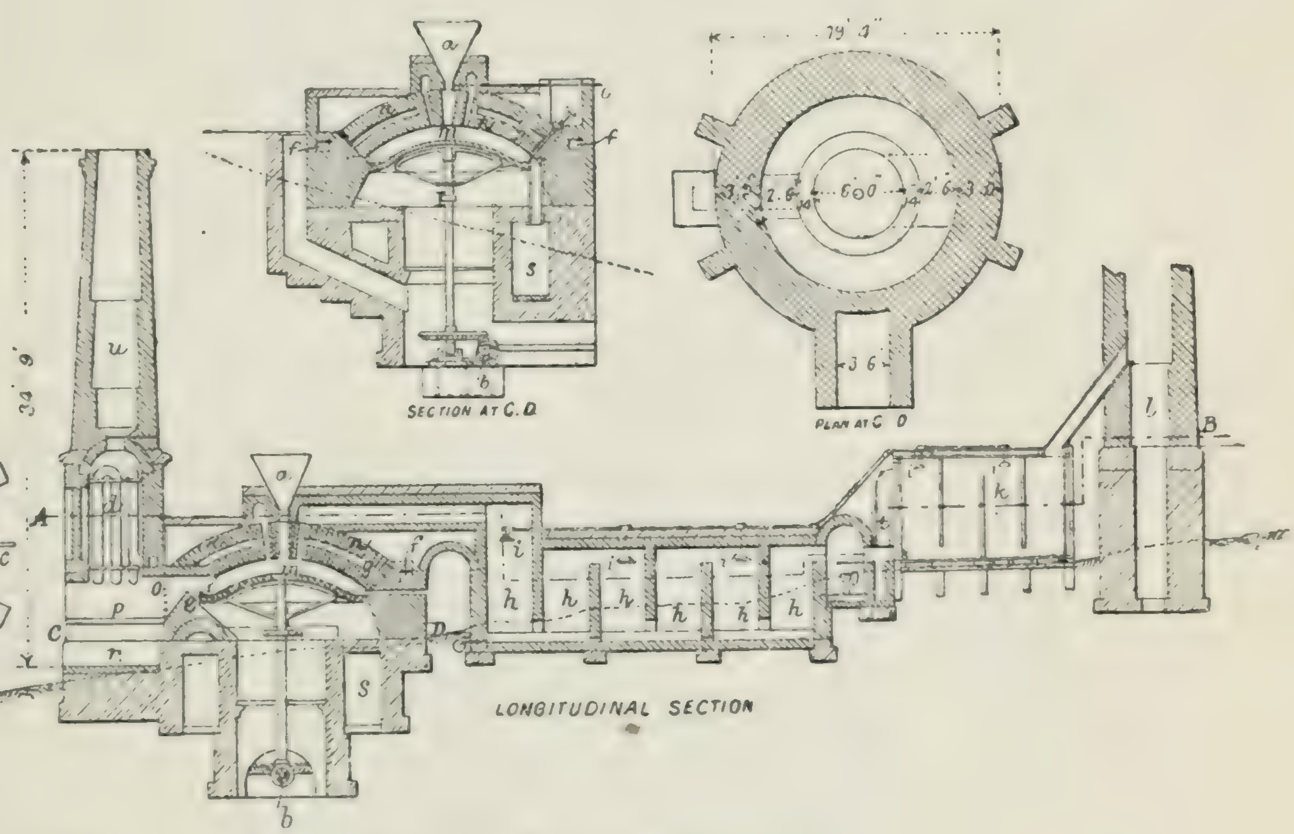




FIG. 120.—SERJEANT AND FLUDE FURNACE.

arsenic). The feed hopper delivers automatically, and can be adjusted to suit the demands of the ore and the speed of the furnace. No fire passes over the ore during roasting, but an abundant supply of heated air is furnished through holes in the crown of the furnace. In this respect, the furnace is a great step in advance. The duration of the roasting can be nicely adjusted to the needs of the ore, and every product is utilised. This last feature, if not yet a pressing matter on most gold-fields, is bound to assume greater importance, both on economic and on sanitary grounds, in the near future, and

no system can be regarded as perfect which entails waste. The mingling of the products of fuel combustion with the sulphurous vapours emitted by the ore acts injuriously, both in retarding the oxidation and in spoiling the gaseous product. On the other hand, this furnace needs as much motive power as the Brunton, and it is difficult to understand how a gradual and steady passage of the ore particles from the centre to the periphery of the hearth can possibly be secured by the operation of gravity alone. The reference letters indicate :—*a*, feed-hopper; *b*, driving gear; *c*, shaft of self-feeder; *d*, hot-air pipes; *e*, fire-bridge; *f*, hot-air flue; *g*, air-chamber in crown of furnace; *h*, condensing chambers; *i*, steam jets; *j*, nitric oxide generator; *k*, leaden sulphuric acid chamber; *l*, chimney; *m*, hearth; *n*, crown; *o*, damper; *p*, fireplace; *r*, ash pit; *s*, roasted-ore pit; *t*, flue to acid chamber; *u*, furnace chimney.

Another modification of the Brunton, coupled with a short reverberatory, has been adopted at the Bunker Hill mine,

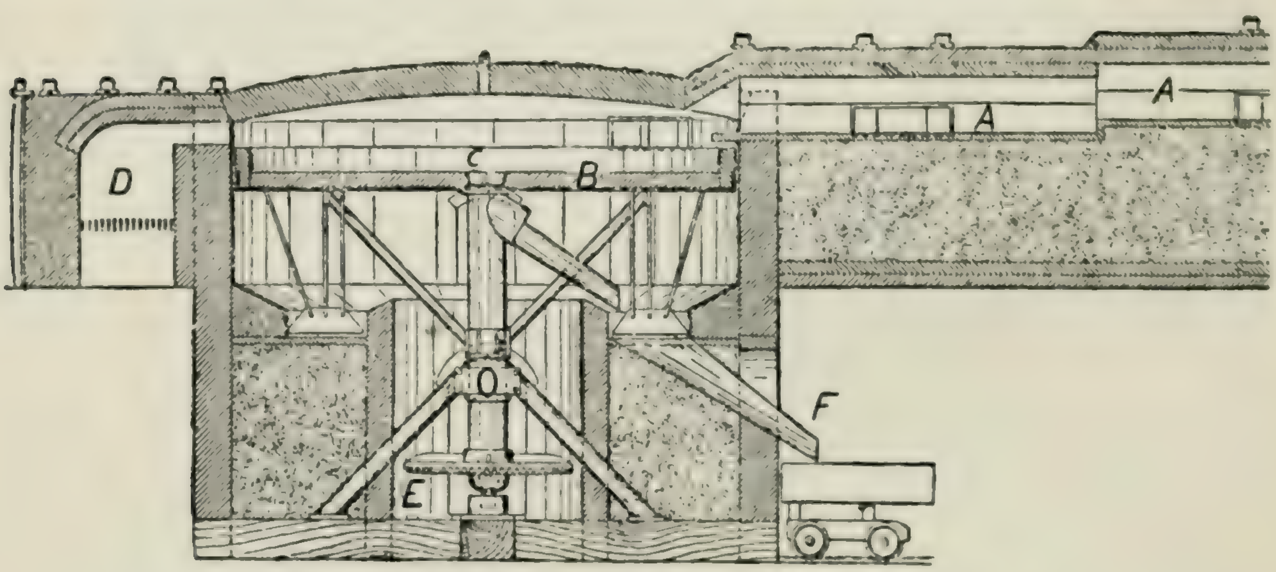



FIG. 121.—MODIFIED BRUNTON FURNACE.

California, and is shown in Fig. 121. The ore, consisting of gold-mill concentrates carrying antimony and lead as well as the usual sulphurets, is worked from the ordinary reverberatories A on to the horizontally-revolving hearth B, which is built of iron lined with firebrick, measures 12 ft. diam., and has a central discharge opening C. Motion at the rate of one revolution a minute is given to the rotating hearth by gearing E beneath. The fire-gases from D have to pass

across the revolving hearth before reaching the stationary hearths. The discharge is at F. The furnace possesses no obvious advantages, its capacity being only 2 t. per diem., with an expenditure of $1\frac{1}{4}$ cords firewood, so that the cost must be high. The fuel employed is pinewood at 25*s.* a cord (= 15*s.* $7\frac{1}{2}$*d.* per ton). Water under 260 ft. head, costing 10*d.* per miners' inch per diem, is used at the rate of 5 m. i. for revolving the furnace and chlorinating barrels, say half each = 1*s.* per ton; while the total labour (including chlorination) amounts to 19*s.* 9*d.* per ton.

The Godfrey mechanical calciner is one of the latest types, and was primarily designed for calcining zinc and lead-zinc ores. While having the usual annular rotating hearth, its novelty consists in placing the rabbling plough in a section of the circle which is open to free air, so that every 2 or 3 minutes the roasting ore encounters fresh air at the moment when it is being stirred. This furnace has been installed at Fry and Everitt's smelting works, Swansea, and is favourably spoken of by them.

# CHAPTER IX.

## CHLORINATION.

THE principle underlying this method of gold extraction is the rapid and complete action of chlorine on gold, whereby is formed a soluble tri-chloride of gold ($AuCl_3$), which can be washed out from the gangue by means of water.

The process may be said to consist of three separate stages :—

1. Impregnating the ore with chlorine to effect dissolution of the gold.

2. Leaching the chlorinated ore with water to remove the gold solution.

3. Precipitating and collecting the gold from its solution.

**The Ore** must first be roasted (pp. 361–409) to render it porous, and effect dehydration of iron salts, or oxidation of sulphides, etc., and a thoroughly complete roasting is absolutely essential ; nothing will make up for an inefficient roast, and it must be obtained, at whatever cost. Poor extractions and great waste of labour and materials ensue if the calcination is neglected.

The chemistry of chlorination has been minutely studied by Dr. T. K. Rose, who points out the great waste of chlorine by imperfect roasting, in that every 1 % of unoxidised sulphur remaining in an ore converts about 9 % of chlorine into hydrochloric acid, and 1 % of sulphur in the form of ferrous sulphate similarly destroys a little over 1 % of chlorine. High consumption of chlorine is due to these causes. Decomposition of the water present accounts for only a very small quantity ; but all sulphides are oxidised to sulphates, with formation of

hydrochloric acid, and, if this is in quantity, some sulphates are converted into chlorides, and sulphuric acid is liberated. When sulphates are present, especially iron or copper sulphates, they must be leached out with water before chlorination is attempted. Metallic oxides, with the exception of the sesquioxide of iron ($Fe_2O_3$), are rapidly attacked by chlorine, and should be dissolved out as sulphates, by adding dilute sulphuric acid, as a preliminary to chlorination. Lime and magnesia are dealt with in the furnace (see pp. 366–7).

Two distinct classes of ore are treated by chlorination :—(*a*) Concentrated sulphurets collected by Frue vanners and similar machines, as produced at the great majority of gold mills throughout the world ; and (*b*) certain ores as mined, notably at Mount Morgan (Queensland) and in the Black Hills (S. Dakota). The gold contents of the former commonly range between 2 oz. and 5 oz. per ton ; and of the latter between ¾ oz. and 1 oz. Examples showing the variation in composition of the gangue are given below and overleaf.

ANALYSES OF CONCENTRATED SULPHURETS.

| — | Eureka and Idaho Mines, Calif. | Washington Mine, Calif. | Black Bear Mine, Calif. | Sonora, Calif. | Port Philip, Vic. | Gympie, Q. | Deloro, Can. |
|---|---|---|---|---|---|---|---|
| Copper | 0·85 | .. | .. | .. | .. | 17·02 | .. |
| Lead | 0·78 | 1·50 | .. | 0·40 | .. | 2·01 | .. |
| Antimony | .. | .. | .. | .. | .. | 3·90 | .. |
| Zinc | .. | 1·34 | .. | .. | .. | .. | .. |
| Iron | 40·65 | 30·85 | 42·05 | 36·54 | 61·25 | 31·41 | 38·00 |
| Arsenic | trace | .. | 21·25 | 8·49 | 6·85 | 0·42 | 42·00 |
| Sulphur | 32·80 | 31·33 | 25·10 | 37·25 | 6·46 | 37·86 | 20·00 |
| Silica | 12·64 | 33·30 | 10·35 | 17·18 | 15·50 | 7·16 | .. |
| Alumina | 0·10 | .. | 0·85 | .. | .. | .. | .. |
| Magnesia | 3·50 | .. | .. | .. | .. | .. | .. |
| Oxygen and loss | 8·65 | 1·67 | 0·38 | .. | 9·90 | .. | .. |
| Gold | 4¼ oz. | .. | .. | 98¾ oz. | 11½ oz. | 12½ oz. | 2½ oz. |

ANALYSES OF ORES AS MIXED.

| — | Black Hills, S. Dak. | Boulder, Colo. | Cripple Creek, Colo. |
|---|---|---|---|
| Silicious rock . . . . . | 89·95 | 89·50 | 63·13 |
| Alumina . . . . . . . | 1·20 | .. | 29·94 |
| Iron . . . . . . . . | 4·26 | 4·50 | 3·66 |
| Copper . . . . . . . | 0·04 | 0·10 | .. |
| Arsenic . . . . . . . | 0·45 | .. | .. |
| Antimony . . . . . . | traces | .. | .. |
| Sulphur . . . . . . . | 0·59 | 2·20 | 2·36 |
| Lime . . . . . . . . | 0·55 | 1·50 | 0·70 |
| Magnesia . . . . . . | .. | 0·25 | trace |
| Combined water . . . | 1·12 | .. | .. |
| Oxygen and $CO_3$ . . . | 1·84 | .. | .. |
| Gold . . . . . . . . | 12 to 15 dwt. | 13 dwt. | 2½ oz. |

After roasting, a ton of concentrates will occupy about 25 cub. ft. This weight is derived from 2800 lb. raw sulphurets, which require about 14 cub. ft. per ton. A ton of sulphurets will weigh 1450 to 1700 lb. after roasting, and will fill about 17½ cub. ft.

At this point samples should be regularly drawn from the ore, say ¼ cwt. from each ton, and bulked at the end of the day; from this bulk, triplicate samples should be taken for assay, after the usual mixing and quartering of the heap.

**The Water** used in chlorination should be filtered free from suspended organic matters, and should contain neither alkaline salts nor ferrous sulphate in solution. Thus mine waters, sea-water, and the water from many springs and lakes is inadmissible till after purification.

**Chlorine** may be generated either in a separate vessel: from salt, manganese and sulphuric acid; or among the ore: from bleaching powder and sulphuric acid. The former plan is practised when the ore is treated in open vats, and the latter when in closed barrels.

These are the recognised sources from which chlorine is prepared, but other means have been proposed.

For instance, in the Pollok process, sulphuric acid is replaced by soda bisulphate, which is a harmless and easily portable salt that may be packed in barrels.

In the Etard process, $5\frac{1}{2}$ oz. commercial potash permanganate dissolved in 100 gal. water is mixed with 20 lb. strong commercial hydrochloric acid ; and in Prof. Black's modification of the same, 6 to 7 oz. potash permanganate dissolved in 100 gal. water is mixed with 12 lb. common salt and 14 lb. strong sulphuric acid.

Many experiments have been also made on the very attractive idea of electrolysing a solution of common salt. Well-designed electrolytic plant, under ordinary commercial conditions, is calculated to yield from decomposition of common salt about 8 lb. chlorine per h.p. in 24 hours. Assuming an average consumption of 4 lb. chlorine per ton of ore (see p. 419), this would mean 50 h.p. to produce the quantity needed for 100 t. of ore per diem, or $\frac{1}{2}$ h.p. per ton. It is thus only where power can be derived from the cheapest sources that electrolysis can possibly compete with the ordinary chemical methods of generating chlorine.

None of these innovations has attained even a moderate degree of commercial success.

When sulphuric acid is not produced on the spot, utilising (as in some instances at any rate) the sulphur burned out of the pyrites, it is transported conveniently in iron tanks or glass carboys.

The introduction of potash permanganate as an excitant in generating chlorine is supposed to avoid any occurrence of free chlorine in the liquors, and thus to obviate delays in discharging the vats after leaching, and to economise chlorine. The process has been tried at Monowai, New Zealand, and at Bethanga, Victoria, but the results do not seem to have been in any way remarkable. The solutions have to be used within 24 hours after mixing. At Monowai, the cost for chemicals was 4*s.* 6*d.* to 5*s.* 5*d.* per ton, so that no economy was manifested over the cyanide process already in vogue.

The addition of chlorides of some of the metals (lead, manganese, etc.) is suggested (and patented) by J. Laudin, on

the ground that "in presence of chlorine, they form higher combinations, which act as transmitters of chlorine, then working *in statu nascenti.*" The assertion is made (June 1898) that experimentally excellent results have been obtained in a "State Laboratory," on a "very refractory ore, not suited to cyanide treatment," the extraction being 71 % as against 45 %, and the saving in chemicals and time about 40 %. But application of the process does not seem to have extended beyond that laboratory as yet.

For the vat method, the chlorine necessary for a charge of ore is usually made in a covered generator *a*, Fig. 122, built of very heavy sheet or cast lead, or of iron lined with lead, the joints of which have been "burned" together, not soldered. This is 20 to 24 in. diam., and 11 to 12 in. deep. The cover is made gas-tight by a water joint. It is raised when necessary by rings *b* on the side. The bottom is made of lead, 16 lb. to the sq. ft. The sides weigh 8 lb. per sq. ft. The generator is placed on a sand-bath *c*, standing either on a perforated arch *d* over a fireplace *e*, or on a steam-chest, and is heated up to about 90° F. A convenient arrangement is to stand the gas generator on bricks in a wooden tub mounted on a truck. Through the centre of the cover a shaft of hard wood *f*, provided with a water joint *g*, and to which a horizontal arm *h* with stirrers *i* is attached, is passed. It has a crank *k* at the top, the object of which is to stir the mixture in the tank after the acid has acted for a certain time. When not in use, this agitator is raised 6 in. The gas is prevented from escaping by

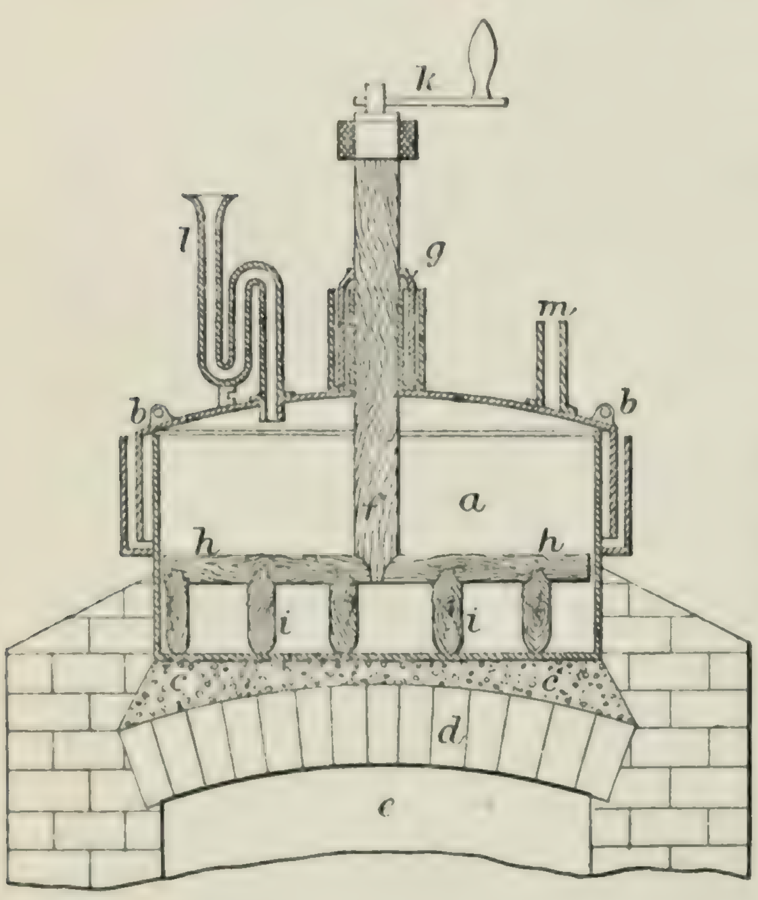



FIG. 122.—CHLORINE GENERATOR.

the hydraulic packing. The usual charge for 3 t. of ore consists of 20 to 24 lb. salt, 15 to 20 lb. manganese binoxide at 70 % (pyrolusite), 30 lb. sulphuric acid at 66° B., and 18 lb. water. When necessary, the acid is carried to 45 lb. and the water to 27 lb. The water is not introduced until the gas is wanted. The cover is generally removed to put in the solids and water, but sometimes there is an opening on the top for the introduction of the charge without lifting the cover, and one in the bottom for removing the spent material. The simplest way of emptying the generator of its exhausted charge is to construct it with a tube leading from the side (at the bottom) through the wall of the water bath, and to sluice out the contents.

Both openings are closed with wooden plugs covered with rags thoroughly saturated with grease to prevent the action of the chlorine. The acid is introduced through a siphon *l*; ½ lb. water is added to every lb. of acid. This charge will be 3½ to 4 in. in depth. The acid is added in pitchers of about ½ gal. at a time. In some works, the acid is placed in a leaden reservoir, whose stop-cock is directly over the siphon, and the acid is made to trickle in. Such a charge will generally run 10 to 12 hours.

When ready to work, rubber tubing, which has been thoroughly greased on the inside, is placed upon the exit pipe *m* in the cover, which is ⅜-in. calibre, and is attached to the entry tube *a* of the wash bottle *b*, Fig. 123, and the exit tube *c*, to which another hose is attached, which connects it with the leaching vat. The wash bottle generally is a large-sized acid bottle or an old carboy, from which the bottom is removed. It is supported in a wooden box *e* lined with lead and filled with warm water, which absorbs less chlorine than cold water. This

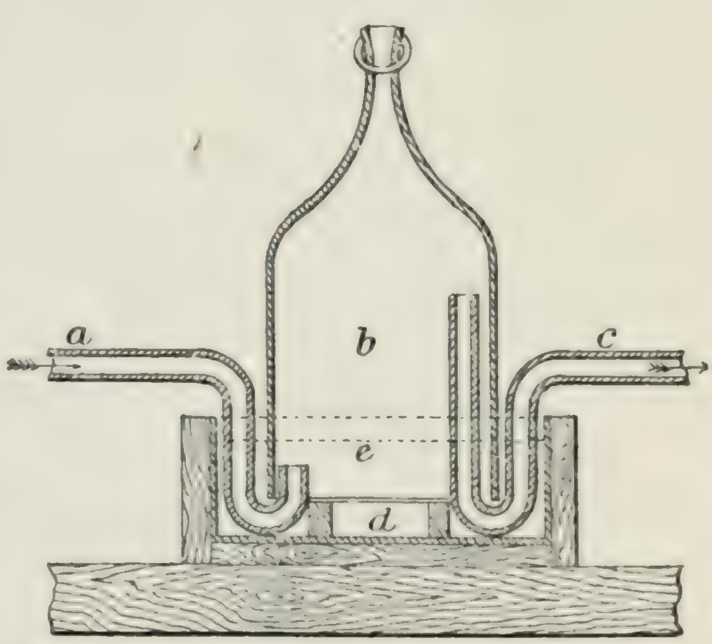



FIG. 123.—CHLORINE WASHER.

water will need changing as often as it becomes charged with excess of acid—probably once a day. The inlet pipes are held in place by a wooden support *d* in the centre of the box. The water in the box stands ½ in. above the inlet pipe. This apparatus is simple, but requires that the pipes shall be fixed so that there can be no danger of their turning over, which they may do if not thoroughly supported. Sometimes a special washing apparatus is made. While a certain amount of hydrochloric acid is thus washed out of the chlorine, that is a matter of much less moment than the service performed by the washer in regulating the supply of gas and rendering it as uniform as possible. In other words, the apparatus is more useful as a receiver than as a washer. The bubbling should be lively, and the washer be filled with greenish gas. When it flags, more sulphuric acid is added, till the allowance is used up. Finally, the temperature of the bath is raised to expel the last of the chlorine. Occasionally the stirrer gets clogged, in which case it must be raised several inches, which can be done without allowing the chlorine gas to escape, as the water-joint is quite long. There will generally be no difficulty in breaking up the charge when the stirrer is elevated a little.

At Deloro, Canada, the chlorine is made from a charge of 40 to 50 lb. chloride of lime and 50 to 60 lb. sulphuric acid. It is forced into a cylinder under pressure at 40 to 50 lb. per sq. in., and falls to 25 to 30 lb. when the chlorination has been made as perfect as desirable. The operation lasts about 2 hours. [Cyanide has now replaced chlorination.]

The chlorine generator is generally placed in the centre of the row of vats, so as to be as near as possible to them all ; but it may be beneath them, or on one side, or there may be several.

At Mount Morgan, where 168,000 to 224,000 t. of ore are chlorinated annually, the chlorine gas is generated from salt, pyrolusite and sulphuric acid in steam-heated stills, and collected in a gasholder, whence it is drawn for saturating water in " scrubbing towers " filled with acid-resisting material, such as old firebrick, clay crucibles, etc. The consumption of this

chlorine water is at the rate of about 175 gal. per hour, costing 11½*d.*, for the oxidised ore, and 700 gal. per hour, costing 3*s.* 10*d.*, for the pyritic ore. The materials consumed in 1900 for treating 268,000 t. of ore were 901 t. salt, 956 t. pyrolusite, and 3308 t. sulphuric acid ; or 7½ lb., 8 lb. and 27¾ lb. per ton respectively. The salt is imported from South Australia ; the pyrolusite (manganese dioxide) is principally obtained from local sources, and averaged 71·24 % available in 1899, as against only 62·8 % in 1898 ; while the sulphuric acid is made on the spot from Japanese brimstone, 860 t. of that article and 112 t. nitrate of soda being used in 1899. The total cost of salt, pyrolusite and sulphuric acid was slightly over 17,000*l.*, or about 1*s.* 6¼*d.* per ton of ore chlorinated. Efforts to cheapen the sulphuric acid and increase its strength are being made by adding Gay Lussac and Glover towers to the plant, and endeavouring to utilise the $SO_3$ generated in the pyrites furnaces.

On the 1½- to 2-oz. concentrates of the Treadwell mills, Alaska, the consumption per ton (unroasted) is 57·6 lb. sulphuric acid, 17·35 lb. manganese, 151·44 lb. salt, and 8·25 lb. scrap iron. The sulphuric acid costs ·85*d.* per lb., the manganese ·75*d.*, and the salt ·22*d.* per lb. Hence the cost per ton for materials is 9*s.* 2½*d.* The contrast between these figures and those of Mount Morgan is most remarkable.

The Delano mill, Colorado, on a telluride ore (see assay, p. 412), uses 10 lb. bleaching powder and 20 lb. sulphuric acid per ton ; the former costs 1¼*d.* per lb., and the latter ⅔*d.* per lb. at the mill, so that materials total about 2*s.* 3½*d.* per ton of ore chlorinated.

One of the Black Hills plants, treating "red" or oxidised ore (see assay, p. 412), employs 10 lb. bleaching powder, costing 1¾*d.* per lb. (1894), and 13 lb. sulphuric acid 66° B., at 1*d.* per lb., delivered ; total cost for materials, 2*s.* 6½*d.* per ton of ore chlorinated. A larger mill in the same district, on a mixture of red and "blue" (pyritic) ores, uses 10½ lb. bleaching powder at 1·62*d.* per lb. (1891), and 26 lb. sulphuric acid at ·85*d.* per lb. ; total cost for materials per ton chlorinated, 3*s.* 3*d.*

The Haile mine, Carolina, operating on non-cupriferous

concentrates carrying 2½ oz. gold per ton (roasted), succeeds with 10 lb. bleaching powder at 1½*d.* per lb., and 15 lb. "commercial" sulphuric acid at 1*d.* per lb., making the cost for materials 2*s.* 6*d.* per ton chlorinated.

At Deloro, Canada, on concentrates (see assay, p. 411), the proportions of chemicals are 6 of bleaching powder to 7 of sulphuric acid, and the actual consumption ranges from 36 to 54 lb. of the former and 42 to 63 lb. of the latter per ton of ore chlorinated.

Tailings from the copper works at Fahlun, Sweden, containing only 1·55 dwt. gold per ton, have been successfully and profitably chlorinated with a consumption of 5·35 lb. bleaching powder costing 4·46*d.*, 7·47 lb. sulphuric acid costing ·89*d.*, and a little plumbic acetate costing ·67*d.*, or a total of only 6·02*d.* per ton treated. But the low prices of the raw materials must be noted—·83*d.* and ·12*d.* per lb. respectively.

On a very refractory concentrate at Brad, Hungary, containing iron pyrites, zinc blende, argentiferous galena, some antimonial minerals, barytes, calcite, and carbonate of magnesia, the sulphur reaching 36 to 40 %, and the bullion contents being about ¾ oz. gold and 4 oz. silver per ton, the consumption of bleaching powder is 22⅓ lb., costing 38·39*d.*, and of sulphuric acid 91 lb., costing 64·28*d.*, or a total of 8*s.* 6·67*d.* The bulk of the sulphuric acid is, however, consumed in leaching out the oxides of iron; and the prices of the materials are excessive—1·72*d.* per lb. for bleaching powder, and ·71*d.* per lb. for sulphuric acid.

At the Ouro Preto mine, Brazil, the charge per ton of roasted ore is 8¾ lb. bleaching powder and 14¼ lb. sulphuric acid, "with enough water to thoroughly moisten the sand" (McCormick), using a barrel; at another Brazilian mine, employing the vat process, the ingredients are 65 lb. sulphuric acid at 66° B., 25 lb. very coarse and impure salt, 15 lb. crude and impure pyrolusite, and 20 lb. water.

With a difficult American ore, 4 t. require three generators each charged with 32 lb. pyrolusite, 36 lb. salt, and 6 gal. sulphuric acid at 66° B.

The average consumption per ton, extending over several years, at a large Californian works has been 32 lb. pyrolusite at 1*d.* per lb. = 2*s.* 8*d.* per ton; 50 lb. salt at ·375*d.* per lb. = 1*s.* 7*d.* per ton; and 60 lb. sulphuric acid at 66° B. at 1¾*d.* per lb. = 8*s.* 9*d.* per ton; total, 13*s.* per ton.

It is always well to employ an excess of sulphuric acid, particularly if the ore contains copper, lead, or lime, as they are thereby prevented from becoming soluble chlorides, and thus wasting chlorine.

From theoretical calculation, the consumption of chlorine in the treatment of gold ores should be exceedingly small, as but little more than ½ oz. of it is needed to convert 1 oz. gold into the soluble trichloride; in practice, however, the bulk of other absorbents of chlorine present is much greater than that of the gold, and these figures are enormously exceeded.

Where bleaching powder is used as a source of chlorine, its quality must be carefully studied. It is very prone to decomposition and, even when kept in air-tight drums, suffers deterioration. The commercial article, usually considered a mixture of chloride and hypochlorite of lime, contains only 20 to 35 % of available chlorine.

In treating ore in closed vessels, an effort is made to secure the advantage said to be derived from using a saturated solution, under which condition the pressure in the vessel is maintained at about 2 atmos. Working at normal temperatures, this is found to involve on the average a provision of about 24 lb. bleaching powder per ton of ore, while at a little over 100° F. half that proportion will produce as much pressure. Apparently in very few mills is this ideal pressure attempted to be maintained, about 10 lb. per ton being by far the most common figure; so that it would seem practical men seek reduction in working cost rather than haste in treatment, which is a commercially sound policy. In fact, as will be shown later on, further economy is gained by abandoning all idea of pressure, and operating with weak chlorine solutions in open tanks, each charge being in hand for a week or more.

The consumption of chlorine is thus seen to vary between wide limits, or from less than 2 lb. to about 18 lb. per ton of

ore, depending upon the character of the ore, and upon the degree of perfection to which the roasting has been carried.

Modern tendencies are certainly towards weak chlorine liquors and prolonged treatment, rather than concentrated solutions and hastened action ; and if chlorination is to survive competition with other processes, such as smelting on the one hand for rich concentrates, and bromine and cyanide on the other for poor ores, this line of development seems inevitable.

**Cooling the Ore.**—After ore has been roasted, it must be completely cooled before chlorine is brought into contact with it, because warm ore is found to require a greater quantity of gas and to afford less satisfactory extraction.

The common method of cooling is to spread the ore on a floor having an area of about 120 sq. ft. per ton roasted per diem. This floor is usually paved with brick, and enclosed by a wall or rim of $\frac{1}{8}$-in. sheet iron about 18 in. high. The spreading and handling involved in this method create much dust (which is to a great extent lost), and cost a considerable item in labour.

At a few large establishments, special provision of automatic cooling arrangements is made. For instance, at Mount Morgan, the ore is passed through iron cylinders 30 in. diam., which have buckets bolted on to the external circumference, and which revolve in a trough of water ; the buckets pick up cold water at each revolution and empty it over the sides of the cylinder, whence it runs back into the trough, a constant inflow of cold water replacing the heated water as it escapes.

In the Delano mill, Colorado, the roasted ore traverses a series of 120 vertical tubes, 4 ft. long and 3 in. diam., immersed in a current of cold water ; here the ore lies for about 20 min., and is drawn away from the bottom, the tubes remaining always full.

A precisely similar cooling plant was installed at the El Paso Reduction Co.'s mill, Florence, Colorado, but after a short time its use was discontinued, in consequence of the difficulty experienced in keeping the pipes and connections open. This was due to the exceptionally bad quality of the

only available water, which created such a heavy deposit of scale as to involve serious delays and costly repairs. The old practice of wheeling the ore in barrows from the furnaces, and distributing it by hand on a cooling floor, has been reverted to.

The Coloradó City reduction works employ a Durant automatic cooler.

At Cripple Creek, the furnaces discharge the ore at bright red heat into an automatic cooler standing 11 ft. high, and occupying only 5½ ft. sq. of floor space, which delivers it at so low a temperature that it can be held comfortably in the hand.

The Argall automatic cooler is used at the Metallic Extraction works, Cyanide, Colorado, and is shown in Fig. 124.

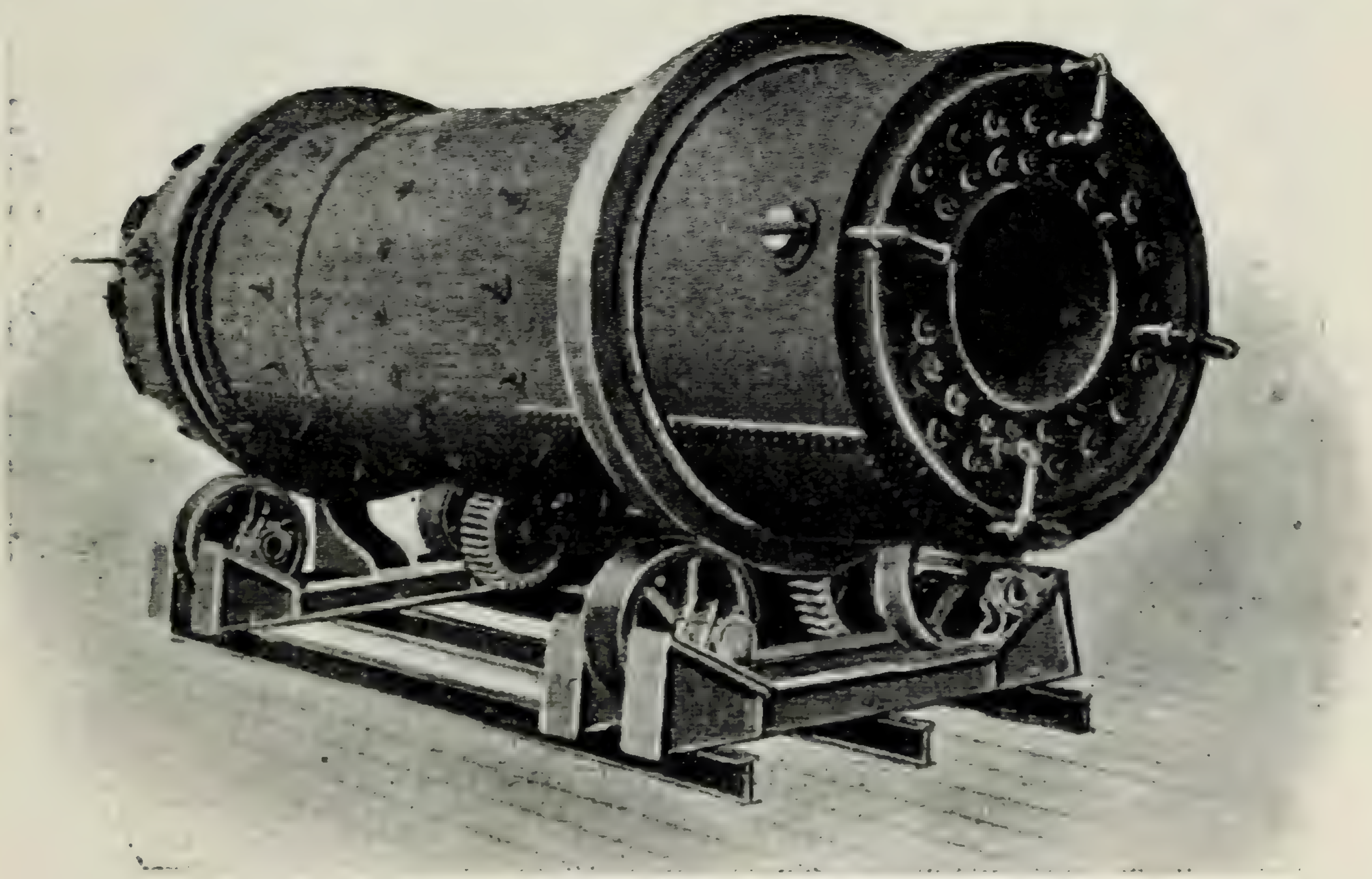


FIG. 124.—ARGALL COOLER.

It consists of a water-jacketed cylinder partially filled with water tubes, set on a slight incline and slowly rotated. The hot ore enters at the upper end and issues at the lower. The largest size cools 300 t. a day.

A trough cooler, made by the same firm (F. M. Davis, Denver) is illustrated in Fig. 125. The hot ore is fed into a

water-lined trough and is moved slowly forward by travelling rakes. One of these in use at the Colorado Ore Reduction works, Elkton, is 90 ft. long and 4 ft. wide, and cools 75 to 100 t. daily.

*Impregnation.*—Endless variety has been introduced into the method of bringing about contact between the chlorine and the ore, and every modification has been termed a new "process" by its author, till the number of these "processes" is

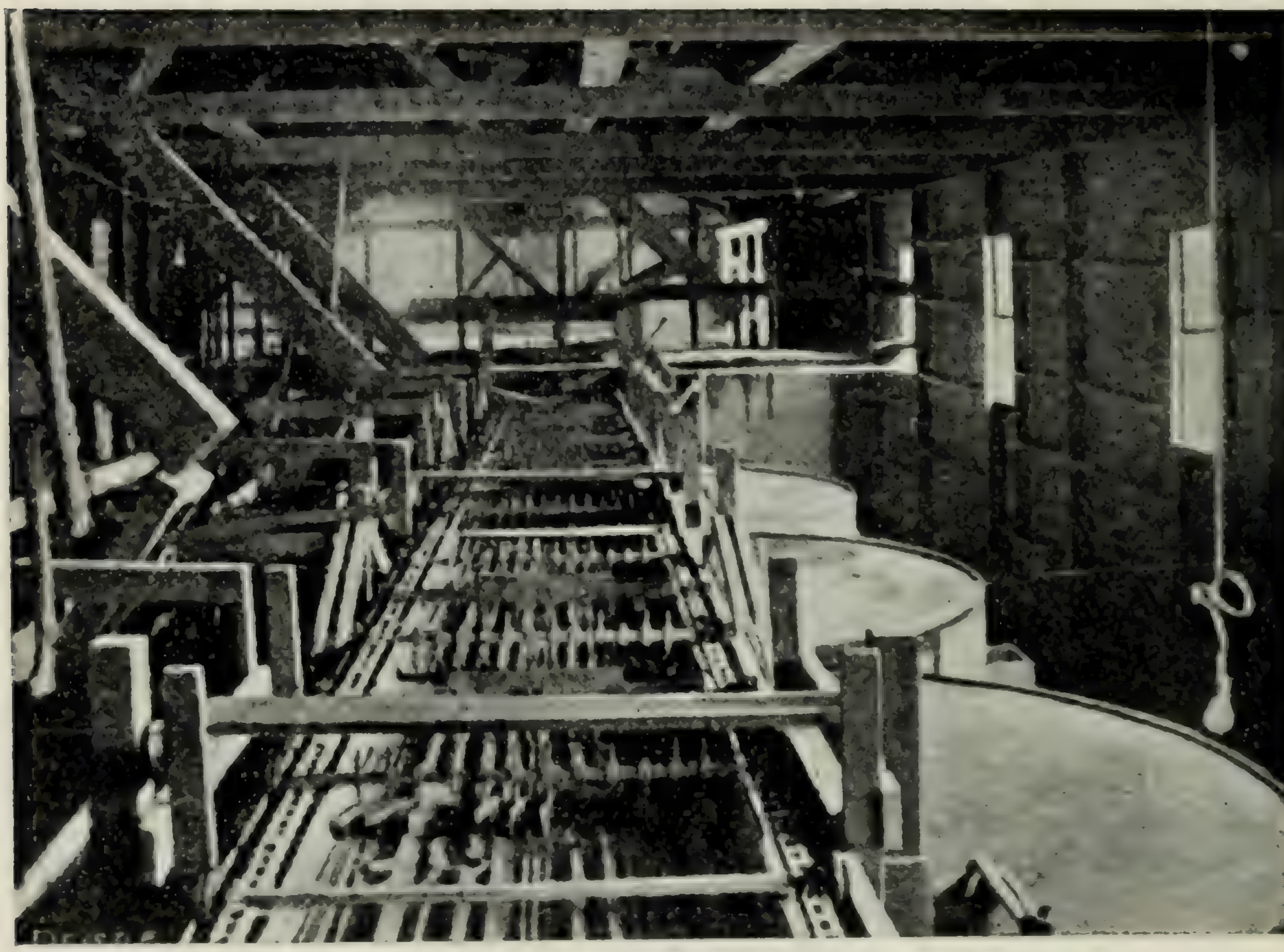


FIG. 125.—TROUGH COOLER.

bewildering. The most essential differences lie in the nature of the vessel in which the operation is performed, and on this mainly depends whether the action is to be free and normal, or under stress caused by pressure or vacuum.

The principal line of demarcation may be drawn between those methods which use a vat or tank, and those which use a barrel. In the former, the chlorine is always separately produced, as already described, and conveyed into the ore either in the gaseous form or in an aqueous solution; in the latter, it

is sometimes admitted as gas, but more often is generated in the vessel holding the ore.

**Vat treatment.**—In its earliest application in California over 50 years ago, chlorination was conducted in vats with lids, on account of the strength of gas used ; and now the tendency is to revert to vats, large ones without lids, and to employ very weak solutions of the gas.

*Vats.*—The covered vat adapted to strong liquors is usually made with a capacity of 2 to 3 t. of roasted ore, and of the form shown in Fig. 126. In California, these vats are built of the best sugar pine, 2 in. thick, 7½ to 9 ft. diam., and 3 ft. deep, and they generally have a batter of 1 in. to the foot. This is done to prevent any space forming on the sides when the ore settles, and thus making points of easy passage for the chlorine gas. The vats are usually placed together in a row, each vat being inclined a little to the front, so as to drain it completely. In front of the vats is a launder *a* to carry off the liquid, which runs through to a sump tank. The vats are thoroughly painted inside and out with 3 coats, usually tar, in some cases asphalt varnish, in others a mixture of coal tar and asphalt melted together and put on hot. This is done to prevent leakage and action of the chlorine. They are repainted once a year.

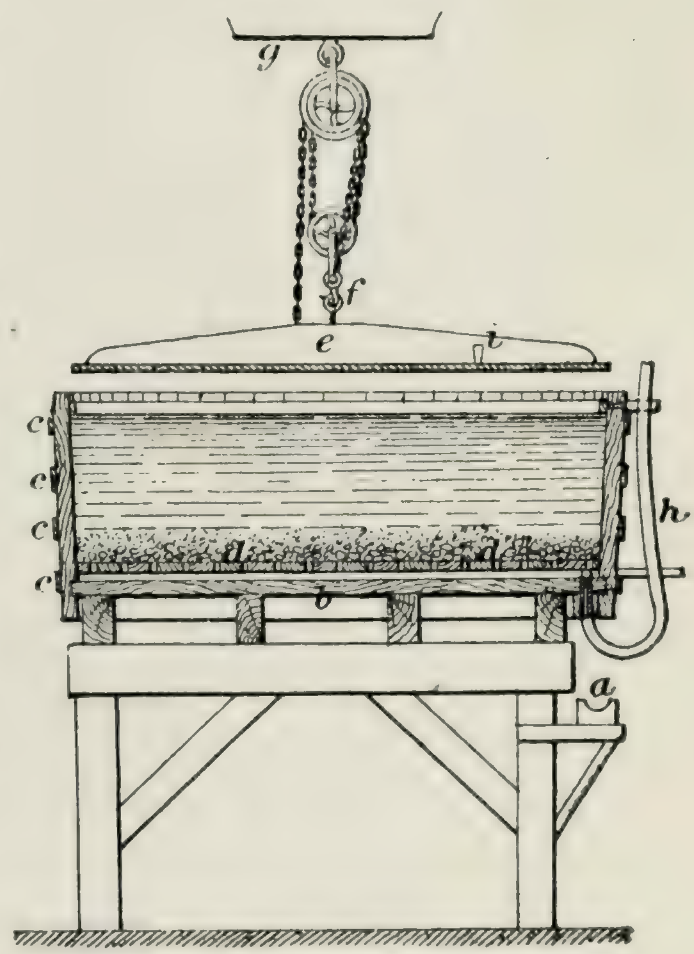


FIG. 126.—CHLORINATION VAT.

At the Providence mill, the bottoms *b* are made of 3-in. grooved planks, which are tongued. The staves are made of 2-in. planks with plain joints, but each joint is lapped on the outside with a batten 2½ in. by 3¾ in. The vat is held together

by 4 iron hoops *c*. With this construction, there is no danger that the iron hoops will be attacked by the chlorine. The diam. of the tubs is at top 5 ft. 9 in., and at bottom 6 ft. 3 in.; capacity 2½ t.

In another Californian mill, they measure 5 ft. 6 in. diam. and 4 ft. 4 in. deep in the clear, and hold 3500 lb. of ore; and again, 6 ft. 8 in. diam. and 2 ft. 6 in. deep, accommodating 3 t.

A Brazilian mill has vats 8 ft. diam. and 3 ft. deep, built of well-seasoned 2-in. plank, tarred inside and out; their nominal capacity is 6720 lb. dry ore, but less than 6000 lb. is found preferable. A gas main conveys chlorine to the series, being a 1-in. leaden pipe with a branch opposite each vat.

In Europe, earthenware and porcelain are substituted for wooden vats, as being non-absorbent; there operations are on such a modest scale that no inconvenience is found from the multiplicity of small vessels.

Despite the various preparations used for coating wooden vats, they become in time saturated with chlorine and with gold; when worn out, they should be incinerated slowly and carefully, and the ashes be fluxed and smelted for recovery of precious metal.

The filter bottom *d* is made of 1-in. boards, placed 3 in. above the bottom of vat and separated by cleats. The depth of the vat above the false bottom is 3 ft. 6 in. These boards are perforated with ¼-in. holes, and are covered with a double thickness of gunny cloth. A cover *e* is made of 1½ in. stuff, tongued and grooved, and reinforced by 4 wide cleats *f* which are screwed on. A groove is cut in the circumference of this cover, into which the vat fits loosely. The cover is sometimes suspended by two iron supports on a rolling truck *g* above. These supports are attached to the cleats *f*.

In some cases, the vats are swung upon gudgeons, as at *a*, Fig. 127. The vat is then held in place by a hook *b*, and the gold chloride is discharged by a stop-cock *c* in the bottom. These vats have a false bottom placed at least 1½ in. above the bottom, in order to allow the exit pipe, which is 1½ in. diam., to enter. This false bottom is pierced with ¼-in. holes, 10 to 12 in. apart. It is sometimes made by placing pieces

of board filled with holes, 2 in. wide and 1 in. thick, 6 to 8 in. apart, over the bottom. These boards rest on cleats several inches apart, which are placed across the bottom of the vat, but do not touch the sides. As, in order to drain completely, the vats must be inclined towards the launder in front, the cleats are made thicker on the front side, so that the false bottom is level. On this bottom, a layer of quartz the size of a hen's egg is spread to a depth of 3 to 4 in. Upon this, smaller pieces half the size are laid 1½ to 2 in. in depth, and so decreasing in size until 6 to 8 in. from the bottom sand is placed. The filter is generally 8 to 10 in. deep. Over this, boards are sometimes placed, set in contact to form a surface to shovel upon when the leached ore is to be removed from the tank. Sometimes gunny sacking is placed over the filter. When the men are skilful, they shovel down on to the filter without disturbing it, but this requires a great deal of practice. This filter remains permanently in the vat until it gets very dirty ; it is then removed to be washed and used over again. The gunny sacking will not endure many days.

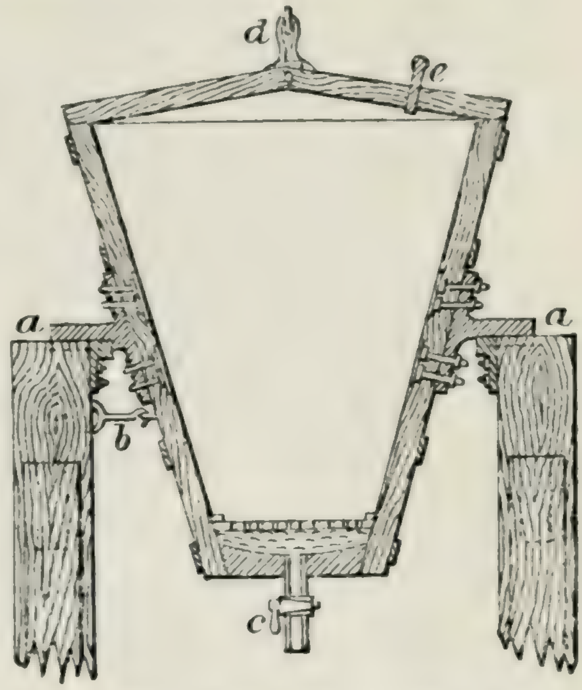


Fig. 127.—Chlorination Vat.

When barrels came into use, about 15 years ago, C. H. Aaron predicted that except in cases where they "yield a notably higher percentage of the gold than by the vat method, the game is not worth the candle. The great advantage of the vat method is that it requires no moving machinery ; there is nothing to break down, and but little to wear out. A set of vats once installed will last a long time if properly cared for, and will consume nothing more than a little coal-tar and asphaltum, or paraffin, from time to time. Mere rapidity in working is of secondary importance ; works of suitable capacity once started, with a constant supply of material to treat, the same daily output is obtained whether each charge takes 3 days or only 3 hours to work. And I see no reason why

vats of 50 t. capacity should not be used in gold leaching as they now are in silver leaching; also they might be arranged to discharge by sluicing out the tailings instead of shovelling, as is also done in silver leaching." This has been realised.

At the South German mill, Maldon, Victoria, the author has seen open wooden vats holding 20 to 30 t. operating with such dilute solutions of chlorine that the vats are unprotected, and not the faintest odour of chlorine comes from the charge. Treatment occupies about a week, and the cost is very low.

Mount Morgan has advanced even further, and the whole of the barrel treatment plant, which was the largest in the world, has been superseded by open tank lixiviation. The vats are excavated pits below ground level, measuring 55 ft. long, 12 ft. 6 in. wide, and 4 ft. deep, lined with cement concrete, and holding 112 t. each. The roasted and cooled ore is trucked to them from bins. The filter-bed is composed of gravel and sand. Leaching is done with weak chlorine water; and the discharge of solid tailings is effected by an automatic scoop travelling on rails on the dividing walls between the vats.

According to E. Hall,* the chlorine is generated in stills, in the ordinary way, from manganese, salt and sulphuric acid, and absorbed in water contained in large cement-lined, covered tanks. The solutions circulate through earthenware pipes, the pumps and valves being nearly all of local design, and representing the "survival of the fittest" in the long experience of the mill in dealing with chlorination. One noticeable feature is the use everywhere of Portland cement where chlorine-bearing liquids are dealt with. It has been found that even the chlorine stills may be made of neat cement. This open vat lixiviation "has reduced costs enormously, and proved an advantage in every way. It simplifies the whole process, and renders it applicable on any scale which may be desired."

*Charging.* — When the ore is completely cooled, it is moistened with about 5 to 10% of water, so that it will just pack or bind in the hand but will gradually fall abroad again when

* En. & Min. Jl., Oct. 7, 1899.

the pressure of the hand is withdrawn ; it is then ready for filling. But the first portion of the charge of the vat is 15 to 20 buckets of dry (roasted) ore, which is spread over the bottom, in order to take up any moisture which may remain there. If it becomes too damp, a sufficient quantity of dry ore is added. Wet ore over the false bottom would prevent the easy passage of the gas through it, and cause an increase in the consumption of gas ; if very wet, the whole quantity of gas used for the charge may not rise more than 2 in. above the bottom of the filter. When it is too dry, the ore is not properly acted upon. If the bottom of the vat is not dried in this way, the lower stratum of ore will take up the moisture and be too damp.

When the rest of the charge is of the proper dampness, it is raked together and charged with a shovel on to a brass wire sieve with 4 to 8 holes to the inch, and falls on to the filter. This sieve has wooden sides and is 3 to 6 in. deep. It either runs upon rollers on two tracks placed at right angles to each other over the top of the vat, or is supported above it by ropes placed at each corner, so as to give it complete freedom of motion. In some cases, it has handles on the sides, so as to avoid the labour of reaching over the front in order to get at all parts of the vat ; in others it has none, and the men are obliged to lean over the top to make the ore fall in the proper position. By moving the sieve backwards and forwards and from side to side, the roasted ore is distributed as evenly as possible, and is every few minutes raked evenly over the bottom by the man who throws the ore on the sieve. The object of the screening is to have the charge porous, to make it equally damp and uniform in size, and to prevent nails and pieces of iron, which would precipitate the gold, from getting into the vat. In some works, the ore is moistened and put into a pile, and is then sieved into wooden boxes, which hold about 50 lb. each, and their contents are carefully poured into the vat in such a way as to be sure that no packing takes place.

The vat is filled in this way to within 2 to 4 in. of the top. It is necessary to leave this amount of space in order to be sure that the whole of the chlorinated ore shall be covered

with water when the leaching commences. If this were not so, the ore would be unequally moist in different parts, and it would be very difficult to get all the soluble gold out of it.

*Gassing.*—When there is about 1 ft. of ore in the vat, the chlorine is turned on. It takes a man 2 hours to charge the vat. He must work rapidly and well, not to be caught by the gas, for by the time the vat is completely filled the chlorine is near the top. On the top, the ore is made highest on the sides and lowest in the centre, so as to be saucer-shaped. It is then lightly pressed with the hand around the whole circumference of the vat, in order to pack it and prevent the free escape of the gas there. The gases have always a tendency to follow the sides of a vat if there is an easy outlet for them.

When the vat is ready, a cover, which is made of flat boards strengthened on top with 6 wooden braces 2 in. wide and 3 in. high, is placed in position. This is coated on the inside with tar, and suspended in many works by means of a chain and pulley blocks, as in Fig. 126; in others, there is simply a ring *d*, as in Fig. 127. Where a number of vats are placed in a row, they are served by a pulley, running on a rail *g* overhead. There are two ways of closing the vat; either a groove is made in the cover which fits over the edge of the vat, or the cover itself is made to fit the inside of the vat, as in Fig. 126, leaving a space between it and the sides of the tub, but resting upon a projection placed there for the purpose. This joint is usually caulked first with rags which have been wetted, and then is made perfectly tight, either with a mixture of spent ore, bran and water, with clay and sand of the proper consistency, with moistened linseed meal, or, as is generally the case, with tough dough, as this is the cheapest material that can be used for the purpose. It is always necessary to keep these joints moist during the process, by placing damp rags over them.

The gas is introduced in some works by a single pipe (*h*, Fig. 126, and *c*, Fig. 127), which enters under the false bottom; in others, there is a pipe on each side, which does not

seem to present any special advantage. The pipe is made of lead, and is introduced through the sides. It is enlarged from the inside, so as to make it perfectly tight, and then flanged. In some works, a leaden nipple, treated in the same way, is used for attaching a rubber hose, and this seems to be the best disposition. A small hole in the cover is left for the escape of the air, and to observe the moment when the chlorine reaches the top. This is done by presenting at the hole a glass rod dipped in ammonia solution ; if chlorine be present, copious white fumes are given off. The hole is then closed with a plug or cock (*i*, Fig. 126, *e*, Fig. 127), but the current is kept up until the charge in the gas generator has been completely exhausted. It is generally passed for 5 to 8 hours, and sometimes 12 hours, the time depending upon the size of the ore. When the gold is very fine, it will take less time than when coarse. To be economical, the operation must be conducted quickly.

After the charge is thoroughly gassed, the chlorine tube is removed, and the ore is left for the gas to act on it for a varying period, depending on the coarseness of the gold particles, but usually ranging between 24 and 40 hours. Thus the period elapsing between first admission of gas and withdrawal of the gold chloride solution is somewhere about 48 to 72 hours or 2 to 3 days.

Temperature is an important factor in hastening the solvent action of chlorine on gold, and the effect of raising it to 122° to 140° F. as compared with a normal temperature of say 60° F. is most marked ; but this fact does not seem to possess any material value from a commercial point of view, as there could hardly be any economy in applying heat to the charge for the sake of shortening the treatment.

*Excess Chlorine.*—When the charge has remained sufficiently long in the vat, the cover is removed. A great excess of chlorine gas completely fills the space between the top of the ore and the cover, and if it were lifted at once there would be considerable escape. This gas may be absorbed by water previous to lifting the cover, but as the gas is retained in solution it makes it difficult to settle the gold in the precipitation

tanks. Aspiration is frequently used to draw off the excess of gas and to collect it in a receiver, for use in a subsequent charge. This is much more rational than allowing it to pervade the atmosphere of the works to the injury of all persons breathing it, as is often done.

*Leaching.*—The introduction of water for leaching the gold chloride must be done with caution. In some works, to be certain of its equal distribution, the cover is lifted and gunny sacking is placed over the ore before the cover is put on, previous to the introduction of the chlorine. The water is introduced by a hole through the cover, falls on the gunny sacking, and is distributed over the ore. A much better way, proposed by Aaron, is to attach permanently under the cover a coil of lead pipe filled with small holes. One end of the pipe comes through the cover so that the water is sprinkled all over the ore in fine jets. In this way the most gold chloride is dissolved in the least amount of water, so that the first solution which collects under the false bottom is very rich, while that above is comparatively poor. In any case, the vat is filled with water until it is about 3 in. deep over the whole surface of the ore, and is allowed to stand until all the gas which is not absorbed by the water has escaped, which takes place in 30 to 40 minutes.

The rubber tube *h*, Fig. 126, attached to the exit pipe below the false bottom, is turned down into the launder *a*, which is of timber coated with tar. These launders are usually 4½ to 6 in. wide, and 3 to 4 in. deep. The solution is allowed to run out very slowly, and water is introduced at the top of the vat as fast as it flows from the bottom. The yellow solution, having a strong odour of chlorine, runs through a canvas filter in a barrel 18 in. diam. and 2 ft. deep; 2 in. from the bottom is a conical wooden funnel, with a rubber hose, 1 in. inside diam., slipped over the small end. The object of the canvas bag is to catch any ore or sand escaping from the vats. Through this tube the solution is carried to the precipitation tanks. The washing is continued until the solution running through shows no trace of gold. At some mills no filter is used; instead the solution is run into tanks 5 ft. diam. to be

settled, and is drawn off to be precipitated. The use of the filter facilitates the operation.

Washing generally lasts 4 to 6 hours. The ore is then allowed to drain. Sometimes all the waters from the last part of the washing and from draining, which are poor, are kept separate, to be used on the next vat of ore. The consumption of water for leaching is commonly about 2 t. per ton of ore, but the use of this quantity must not be regarded as an assurance that the action is completed. Any curtailment of the process short of practically absolute exhaustion of the gold liquor is of course utter folly, as it means a definite and avoidable loss without a corresponding benefit of any kind. There is no reason why leaching should not be continued till the effluent water shows no more than 3 gr. per ton, or about 1 part in 5,000,000, which can be readily tested (after passing the turbid liquor through filter paper) by degree of coloration produced on adding the filtered liquor, at boiling point, say 1 litre in volume, to 5 or 10 c.c. of saturated solution of stannous chloride, and agitating.* If the stannous chloride is freshly prepared, or a piece of metallic tin is kept in it, a brown precipitate is afforded; otherwise, in the presence of the stannic salt, a purple coloration ensues. Ferrous sulphate is commonly used as a test; but it is not nearly so delicate, and is much obscured or hindered by the presence of copper salts.

At Bethanga, Victoria, an unusual course is followed. The roasted and moistened ore is charged into 16-t. vats. Gas is generated by putting lime chloride and sulphuric acid with water into 2 contiguous vats, joined by a yoke at bottom. On opening a cock, the liquors meet, and a saturated solution of chlorine flows into the ore vat. When it lies 2 to 3 in. deep over the ore, the outlet is opened, and the solution is continuously drawn off. No standing of solution on the ore is permissible, because the excess of sulphuric acid present forms ferrous sulphate with the iron in the ore, and causes premature precipitation of the gold. The extraction is said † to be 90 %,

* Dr. T. K. Rose, Chem. News, lxvi. 271 (1892).

† E. R. Field, Trans. Inst. M. & M., viii. 135-6 (1900).

but it is admitted that much gold is found in the copper which is subsequently thrown down by scrap iron.

*Discharging Vats.*—When the ore has drained dry, it is carefully sampled for assay before the contents are disposed of.

Removal of the solid tailings or residues from the vats is effected in several ways. Very commonly, shovelling into handy trucks for conveyance to the waste dump is in vogue, though costly. Sometimes this is avoided by having the vat on wheels, so that it can be rolled to the dump heap and there emptied. It requires some care to remove the ore from the top of the filter so as not to injure it. The labour is much reduced when the vat is suspended, as it can be turned over, and the ore in it shot out. In other works where the vat is suspended, when turned down its contents are washed out into a sluice by a hose.

At Mount Morgan, the 100-t. cement tanks are discharged by an automatic scoop carried on rails running between them.

The vat when empty is ready for the next charge. Every day one set of vats is charged with gas, a second remains in gas, and a third is filtered and emptied. When there are 3 or more impregnation vats, one set is filled every fourth day. Filtering and emptying are done every third day. When there are 6 or more, several are treated at a time.

**Barrel treatment.**—Chlorination in rotating barrels is as old as the vat method, but having been invented in Europe, where the scope for operations is limited, little was heard of it. Fifteen years later, it was re-invented in Australia, and after the lapse of another 20 years or so, made its appearance in the United States. Since then it has been pushed to the front, and many modifications have been introduced. Its distinctive features, however, have not been changed. They consist in (*a*) generating the chlorine among the ore, so that it may act in a nascent condition, and (*b*) ensuring contact between the gas and the gold particles by a continuous agitation and commingling produced by revolving the barrel. As compared with vat treatment, the advantages and disadvantages appear to be as follows :—

1. While nascent chlorine may be rightly credited with unusual activity, whereby the duration of the treatment is shortened, this is only an offset to the limited capacity of the vessel, and the one may be regarded as balancing the other.

2. Mere speed is not worth attaining, unless it brings about an economy in cost or an increase in extraction.

3. The sources of chlorine used in the barrel method are more costly than those employed in vat treatment, and they introduce into the charge a sediment of sulphate of lime which gives some trouble in its subsequent removal.

4. The mobility and handiness of the barrel admit of the ready adoption of automatic means of charging and discharging, which contrast very favourably with the arrangements ordinarily in vogue with vats; but there is no reason whatever why the filling and emptying of the latter should not be brought down to the same low cost; and it has already been done in the most progressive works (see p. 432).

5. Comparing capacities, the barrel is much more expensive in first cost than the vat, and it suffers much greater wear and tear, thus demanding an increased outlay for up-keep.

6. The motive power required to rotate the barrel necessitates additional expenditure on plant, fuel, and labour.

7. The one real advantage of the barrel appears in the case of ores carrying coarse gold, or a large proportion of silver, the movement and friction assisting materially in presenting fresh surfaces to attack by the chlorine—especially in detaching the film of silver chloride, which is apt to envelop the gold particles. But it is a moot point whether in most cases smelting would not be found more profitable for such ores. It is certainly a very significant fact that the great Treadwell mill, in Alaska, with 540 stamps, and enjoying exceptional advantages for doing its own work, has shut down its extensive roasting and chlorinating plant, and now sells its concentrates to an outside smelter; and several of the Transvaal mills, also under American management, have followed the same course.

*Barrels.*—An example of the modern chlorination barrel, as made by the Gates Iron Works, is shown in Fig. 137. The

shell is of sheet steel, with heavy cast-iron ends ; two charging doors are provided, and a lining of sheet lead (which must be "chemical," i.e. free from impurity), varying from 18 to 24 lb. per ft. Motion is attained by spur-wheel and pinion as shown, or by pulley.

The huge barrels (Fig. 128), 17 × 7 ft., and holding 15 t., made by F. M. Davis, Denver, for the Economic Gold Extraction Co., Victor, Colorado, are too heavy to be carried by trunnions attached to the heads, and are provided with tyres *a*

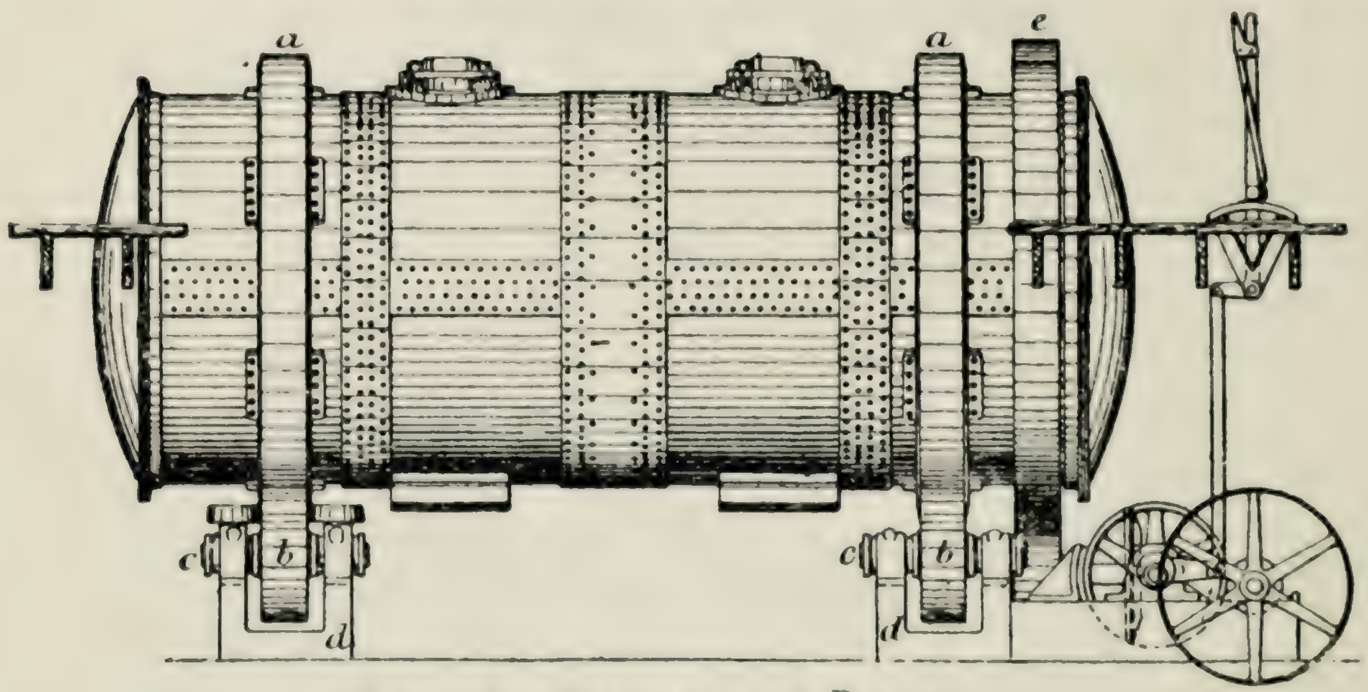



Fig. 128.—Chlorination Barrel.

resting on carrier-rollers *b*. The roller shafts revolve in ball-and socket-boxes *c*, which are supported on massive cast-iron sole-plates *d*. The barrel is driven by gear *e* attached to the shell at one end.

Some sizes, capacities, and weights of barrels are as follows :—

CHLORINATION BARRELS.

| Length. | Diameter. | Capacity. | Weight. |
|---|---|---|---|
| in. | in. | t. | t. |
| 44 | 40 | 1 | 3 |
| 60 | 42 | 1 to 1¼ | 4½ |
| 72 | 60 | 3 | 7 |
| 96 | 60 | 5 | 12 |
| 144 | 72 | 9½ | 20 |
| 204 | 84 | 15 | .. |

In the earlier forms, the trunnions of the barrels were made hollow, so that gaseous chlorine separately produced in a generator might be forced in under a pressure of 30 to 40 lb. per sq. in. The cast-iron heads, which were riveted to the $\frac{1}{4}$-in. boiler-iron shell, were cast with a long trunnion on them; this was turned true, to act as a journal-bearing and driving-shaft, and was cored out, with a stuffing-box and gland fitted to the end.

Through the trunnion is passed the "goose-neck," an iron pipe covered and lined with lead, and turned smooth and true where it traverses the stuffing-box and gland. Inside the cylinder it is bent up, and the top end, which just clears the shelves, is turned in the direction in which the cylinder revolves. The goose-neck is held stationary in an upright position by clamps; its object is to be able to draw off or pump the gas into the cylinder while it is in motion. When a pressure-gauge is attached, the variation of pressure can be noted.

Manholes are provided in the shell for repairs. In the manhole cover is an opening for charging and discharging the pulp, fitted with a light and strong lead-covered lid, arranged to be easily and quickly removed.

Several lead-covered shelves are bolted to the inside of the cylinder for the purpose of stirring the pulp thoroughly.

All joints in the lead lining must be made by autogenous "burning," as in sulphuric acid chambers; no solder is permissible, as the tin contained in it is very rapidly attacked by chlorine.

The apparatus employed consists of a hydrogen gas generator or "burning machine," as it is commonly called, an "air vessel" or portable bellows, some rubber tubing, and a set of gas-cocks and jets. The hydrogen generator is shown in Fig. 129; *a* is an air-tight leaden cistern, having a perforated shelf *b*, and an opening *c* in the top; *d* is another leaden cistern with a perforated shelf *e*. A pipe *f* connects the cisterns *a d*, passing through *a* as far as the shelf *b*, which it just perforates. The hinged and rubber-lined cover *g* being turned back, the cistern *a* is filled with sheet-zinc cuttings, and the

cover is closed. Diluted sulphuric acid, say 1 qt. of acid at 66° B. to 1 gal. water, is poured into the cistern *d*, and finds its way through the pipe *f* into the bottom of the cistern *a*, rising through the strainer *b*, and surrounding the zinc. The

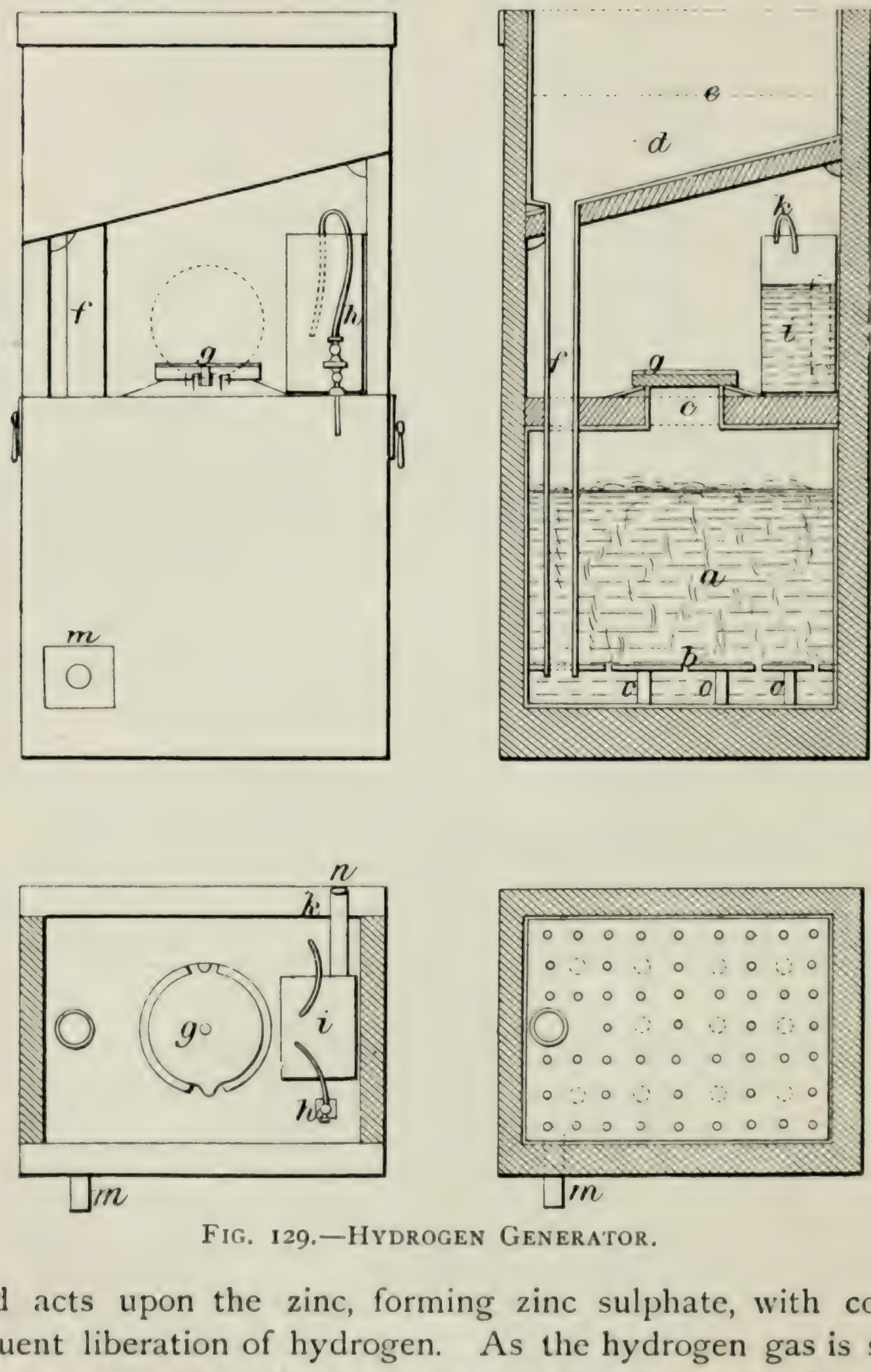



FIG. 129.—HYDROGEN GENERATOR.

acid acts upon the zinc, forming zinc sulphate, with consequent liberation of hydrogen. As the hydrogen gas is set free, it passes through the cock and pipe *h* into the leaden vessel *i*, partially filled with water, and, traversing the water, it

becomes purified, and escapes at the pipe *k*; *m* is the pipe through which the generator is emptied of acid when the gas

FIG. 130.

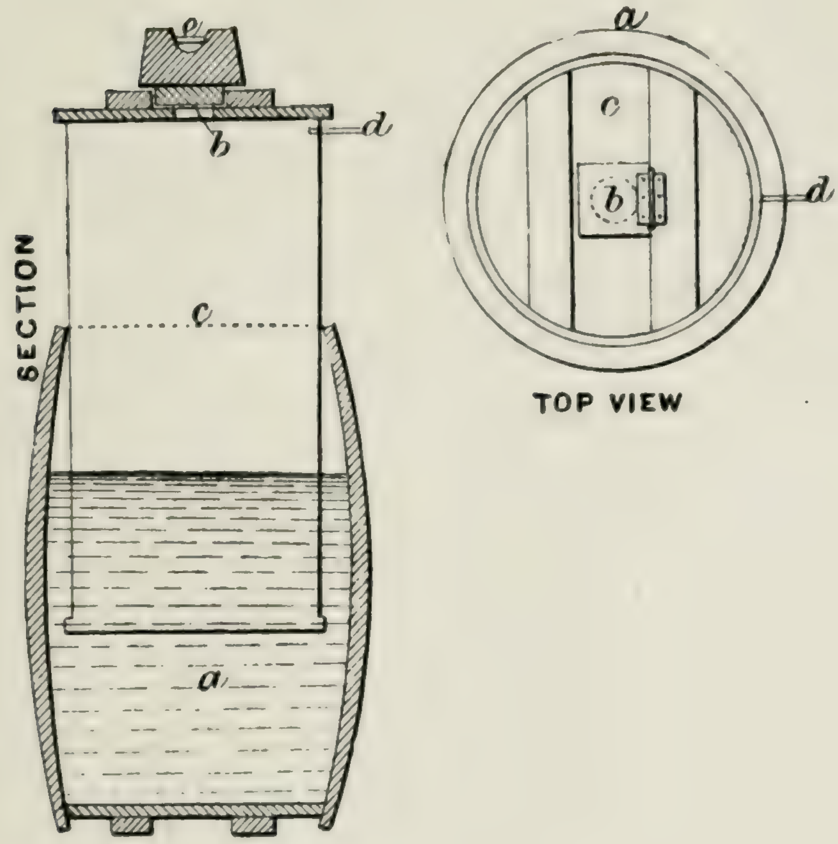
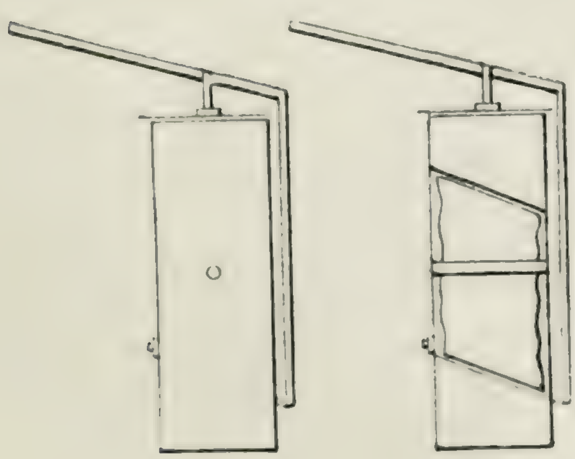



FIG. 131.


FIG. 132.



FIGS. 130 to 132.—AIR VESSELS AND TUBES.

is no longer required. The vessel *i* may be removed from its place by unscrewing the nut close to the cock on the pipe *h*, and may be filled with water or emptied through the pipe *n*.

The pipes *m* and *n* are plugged with corks ; *o* are short pieces of pipe supporting the shelf *b*, to which they are attached.

The air vessel consists simply of a wooden cask open at the top, containing a cylinder of zinc, with a closed top, having a hole and cover in the centre, as shown in Fig. 130 (scale ½ in. = 1 ft.). The cask *a* is partially filled with water, the cover *b* (which is coated underneath with sheet rubber to make it shut close) is opened, the cylinder *c* is raised, and the cover is closed again, preventing the escape of air from the cylinder except through the small pipe *d*. A weight *e* is placed on the top of the cylinder to keep the cover *b* firmly closed, and give force to the current of air issuing from *d*, the weight being conveniently represented by a ¼-, ½-, or 1-cwt., according to the pressure of air required.

A small bellows, Fig. 131, is sometimes used by plumbers for obtaining a supply of air. It is more portable than the air vessel, but cannot be used without an assistant to work it.

Rubber tubes *a b* (Fig. 132) connect the gas generator and the air vessel or bellows with a pair of brass cocks and breeches-pipe *c*. The gas and air, being admitted through these cocks, unite in the tube *d*, and, passing through the brass pipe *e* and jet *f*, may be ignited, and produce an intensely hot flame, by which leaden sheets may be joined without the aid of any flux.

The lead to be burned must first be scraped bright, and where a strong seam is required, strips of clean lead are run on in the manner of solder. But it is essential to success that all the surfaces to be subjected to the flame be bright and dry, and that no moisture be sufficiently near the seam to be drawn into it by the heat. Several jets are in use, with holes of various sizes, for procuring a large or small flame, according to the special requirements of the work in hand ; and the intensity of the heat is also regulated by the proportions and quantities of gas and air admitted through the cocks. As it is imperative that the flame should not be subject to sudden variation, little brass tubes are fitted to the nozzle to guard the flame from air currents when working out of doors or in draughty places.

The pumping in of gaseous chlorine was found to be a matter of great difficulty, and was soon abandoned in favour of generating the chlorine in the barrel. It then became necessary to keep the chlorine-forming materials apart until the barrel had been closed, to prevent the inconvenience and loss of escaping gas. This led to the provision of a leaden pocket, burned to the lining, and calculated to hold about 100 lb. sulphuric acid per ton of ore in the charge. Thus contact between acid and bleaching powder was postponed until the barrel revolved and spilled the acid.

Some 1-t. barrels erected at Mount Morgan were made of wood, $5\frac{1}{2}$ ft. by $3\frac{1}{2}$ ft., with 3 in. sides and 4 in. ends, well braced with iron, and capable of withstanding a maximum pressure of 20 lb. per sq. in.

Curious misapprehensions arose on the question of pressure, and there came to be much confusion between the pressure caused by an excess of chlorine gas, which accelerates its action, and pressure created by pumping in air or water, which has no useful effect at all. Thus we find one of the largest chlorinating works in Victoria forcing air into the barrel till the gauge registered 15 to 30 lb. per sq. in.; and an English company was heavily capitalised to work a patent process using water at 100 to 120 lb. per sq. in., which failed utterly.

On the other hand, it became evident to careful observers, that the importance of pressure had been much exaggerated, and that quite sufficient for practical purposes was gained by generating the chlorine inside the barrel. Leakages and delays caused by wearing out of goose-necks and collapsing of linings through injudicious pressure, often without any chlorine at all, led to a complete reversal of opinion, and Adolph Thies set the example of closing the trunnion, making it solid, and using merely a highly saturated chlorine water, with a lead valve connected with the lead lining to indicate the pressure of free chlorine in the barrel. This is the foundation of modern barrel practice.

While the barrel adopted by Thies is of small dimensions and accommodates only 1 to $1\frac{1}{4}$ t. of ore at a charge, in

recently built mills there is a tendency to large capacities. Thus at the Delano mill, Colorado, erected in 1897, the barrels hold 5 t. each, and measure 8 ft. × 5 ft. inside, the shells being of ½-in. steel and the ends of cast iron, with 12-in. solid trunnions; the lead lining is 18-lb. for the sides and 24-lb. for the ends; the single charging door is 16 × 12 in., and there are 4 connections (2 in.) for pipes. The Colorado City works has 12 × 6 ft. barrels, holding 9½ t. of ore, the shells of ⅝-in. steel and the heads of heavily ribbed cast iron, designed to withstand a pressure of 100 lb. per sq. in.; they are lined throughout with 24-lb. lead. The barrels of the Economic Gold Extraction Co., Victor, Colorado, measure 17 ft. long and 7 ft. diam.; while the new works of the Economic Co., at Eclipse, Colorado, have a barrel capacity of 30 t.

The advantages to be derived from these capacious but heavy barrels are certainly not manifest. They must be very cumbersome and costly to erect and control, and their heavy charge must entail more than usual wear and tear, while cessations for repairs mean a greater curtailment of output than with smaller vessels.

Sometimes the lead lining of the cylinder of the barrel is burned on to that of the ends, but there are objections to this on the score of inconvenience for effecting repairs inside the barrel, and greater risk of injury to the lead lining itself. It is better to let the lining of the cylinder and the heads respectively be quite distinct; a sufficiently tight joint is made by clamping the head very forcibly to the flange bolted on the cylinder, the heavy lead lining of the head forming the packing. Thus the head is free for removal at any time that access to the interior may be necessary.

*Charging.*—For convenience in subsequent operations, it is desirable that both solid and liquid contents of the barrels, after treatment, shall get away by gravitation, and for this reason it is the custom to erect the barrels on a floor considerably above the ground. Hence the roasted and cooled ore has to be elevated to them, and, to facilitate charging, the elevating is carried to a higher floor, where bins are installed for reception of the ore. In some installations there is a

single bin delivering to a truck, which, running on rails, and passing over a weighbridge, delivers a precise quantity of ore to each barrel. In other cases, the hopper over each barrel is of a size to hold a single charge, and is supplied through a shoot leading either from a bin or from the elevator. Connection between the mouth of the barrel and the hopper or shoot is best made by a canvas pipe of sufficient capacity, as this admits of easy disposal out of the way of the revolving barrel when not in use, and can be introduced so far into the barrel as to minimise the creation of dust when in use. Too much damping of the ore is not desirable, or it may clog in the elevators, shoots or hoppers.

The order in which the several constituents of the charge are admitted to the barrel varies.

When an acid pocket is furnished in the barrel, the usual practice is to mix the ore and the bleaching powder together more or less in the hopper feeding the barrel.

The requisite water, computed to produce an easily flowing pulp, is first run in. The quantity will vary with the ore, more being required for concentrates ground to a fine powder and carrying ounces of gold per ton, than for silicious ores crushed only to a coarse mesh and assaying fractions of an ounce. Perhaps an average normal figure would be about 120 gal. per ton in the former case, and 80 gal. in the latter. The volume having been determined for any particular ore, it will not vary much, and the easiest way of apportioning it is by a float or other gauge placed in the barrel. The supply is drawn from an overhead cistern and conveyed by means of a pipe and tap suitably arranged. It is an excellent plan to have the water-supply abundant and at the top of the building, so as to be available in case of fire, as well as for legitimate purposes.

Immediately following the water, the full charge of ore and bleaching powder is admitted. Then, taking care that no ore or bleaching powder has fallen into it, the acid pocket receives its dose through a funnel, and the lid is secured ready for rotation of the barrel.

In many instances, however, the acid pocket has been dispensed with as an unnecessary adjunct, and it is found that, by

adding the sulphuric acid immediately after the water, it to a great extent sinks and forms the lowest stratum. The ore is next introduced without excessive disturbance, and finally the bleaching powder is placed on the top. If the operations are performed with reasonable speed and smartness, the lid may be fixed before any of the acid will have soaked up through the ore and reached the bleaching powder, and no inconvenience will result. At the modern Colorado City works, the acid and bleaching powder are carried from the store-room to the barrels in buckets suspended from two-wheeled crawls running on overhead tracks. The acid may easily be run from a lead-lined gauge-tank by a leaden pipe, and thus all handling of it can be obviated.

The usual practice, as will have been understood, is to introduce the full dose of chemicals for the charge at once. But Thies claims to have found it advantageous in some cases to split the dose, adding the second half after the barrel has been revolved for 3 to 4 hours, and then giving a further 2 to 3 hours' rotation.

*Rotation.*—Complete and searching action of the chlorine, generated when the acid and bleaching powder come into contact, is secured by slow and continued rotation of the barrel. Toothed gearing or friction gearing may be used for producing the motion, provision being made for controlling and stopping it with precision. Belting is most undesirable, being less economic of power, much in the way, and subject to excessively rapid decay in the atmosphere tainted with chlorine. The speed varies as a rule between 4 and 20 rev. a minute, the lowest figures being applicable to the largest barrels. No greater speed than the circumstances demand is advisable, as it involves increased consumption of power and wear and tear. The duration of the rotary movement is prescribed by the nature of the ore, being less as the comminution of the ore has been greater, in general terms, though special cases may demand special treatment ; usually it ranges from 1½ to 6 hours. During that period, repeated test is made—by opening the small valve provided, and applying the ammonia indicator (p. 429)—that chlorine is actually present in

excess. Should it not be so, rotation is useless until the deficiency is made good by stopping and opening the barrel, and introducing a supplementary dose of acid and bleaching powder in due proportions.

The power required to maintain steady rotation of the barrels has never been exactly computed, probably because the work is done by the same motors as operate the milling and elevating machinery. Moreover it is intermittent. It would seem that ample margin would be provided by allowing $\frac{1}{5}$ to $\frac{1}{3}$ h.p. per ton of ore treated per diem.

*Excess chlorine.*—As in the vat process, when the act of chlorinating is complete, an excess of chlorine remains unabsorbed. It is disposed of in various ways. At works which do not claim to be economically managed, the gas is either allowed to freely escape and mingle with the atmosphere inside the building, or is drawn off by a vacuum-pump and hose and ejected into the outside air. The latter plan minimises the nuisance created, but is just as wasteful as the former. It is quite a rare thing for the gas to be conserved for future use by passing it into water, as the barrel process relies on freshly-generated chlorine in the nascent condition; but it is difficult to understand why such a chlorine solution should not replace ordinary water to some extent in charging the barrel, and permit a corresponding reduction in the amount of bleaching powder used.

Often the surplus gas is not dealt with at this stage, but is carried away in solution by the leaching water introduced into the barrel. In that case, its elimination takes place just before precipitation, it being a necessary preliminary to that operation, and technically known as "killing." [A description of it is introduced here though not in proper order of sequence of events.] The most common practice is to force in a stream of sulphurous acid ($SO_2$), by burning brimstone in a closed vessel and supplying it with air from a compressor or pump, until all the chlorine existing in a free state is converted into hydrochloric acid. A flexible (rubber) hose is used, lying on the floor of the precipitating tank, and the pressure causes sufficient disturbance of the hose to bring about thorough agitation and

chemical combination. Another method is to blow air through the solution, thus mechanically expelling the chlorine. Again, in either event, the converted or ejected gas is totally lost. A sulphur dioxide generator is shown in Fig. 133. It is a cast iron tank with air-tight handhole plate, easily removed, and a sulphur-burning tray with cover plate, so arranged that the compressed air admitted is compelled to pass over the surface of the burning sulphur. One suitable for a 100-t. mill weighs 1400 lb. and costs in America 80*l.*

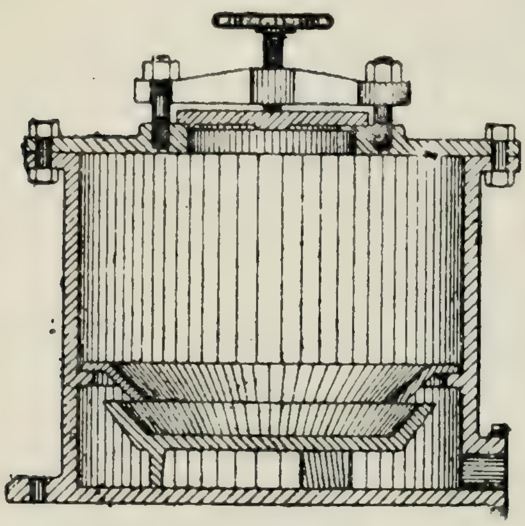


FIG. 133.—SULPHUR DIOXIDE GENERATOR.

*Leaching.* — The leaching or washing out of the gold trichloride solution from the solid contents of the barrel is performed in several ways. The old custom was invariably to discharge the whole contents of the barrel, solid and liquid together, into a separate leaching or filtering tank, the barrel being immediately released for a fresh charge. The newest method is to do the whole of the washing and filtering inside the barrel itself, which is then equipped with a special internal filter. Between these two extremes there are various modifications, the work being more or less done in both the barrel and the tank.

Leaching tanks are generally shallow vessels, made of wood, sometimes lead-lined, but often only tarred or paraffined ; 6-lb. lead will suffice, and should endure for several years. They may be round or square, and of any desired dimensions. Their essential feature is the filter, which consists of a bed several inches thick of quartz pebbles, small stones, and sand in successive layers, the coarsest being the lowest stratum ; sometimes perforated glazed tiles replace the quartz pebbles. The area of this filter is computed to accommodate the charge of solids delivered to it in such fashion that no greater thickness than about 4 in. shall lie on it, in order that a thoroughly free leaching action may ensue. The precaution is taken to cover the filter with a flood of clean water, admitted by a

pipe from below, so that the pulp issuing from the barrel is well distributed, and prevented from making violent impact on the filter, which would disturb it, as well as from packing unevenly and causing imperfect filtration.

The solid ore, when washed entirely free from gold solution, is discharged from the tanks, by manual labour, into trucks for ultimate disposal ; operations are somewhat simplified by laying gunny sacking on a wooden framework above the filter-bed, but the endurance of such a material is very limited, and it must be carefully preserved and incinerated for recovery of any gold which may have been arrested or precipitated by it in the absence of free chlorine. Wooden slats are sometimes used to maintain an even surface and facilitate shovelling.

The passage of the gold liquor and the wash waters through the filter-bed is often facilitated by vacuum-pumps, which pass them on to the settling-tanks. This application of a vacuum is considered in some quarters to be a detriment rather than an advantage, as the effect of the atmospheric pressure on the surface of the mass is not confined to the liquid portion but manifestly acts equally on the solids, compressing them and tending to close all channels of escape for the liquor. In this connection, a remarkable statement is made by Dr. Rose * as to the "effect of the use of increased pressure of air applied directly on the surface of the liquid," in vats capable of sustaining an internal pressure of 100 lb. per sq. in., whereby it "was found that ores, which entirely prevented the passage of water through them, even after a vacuum of 20 in. of mercury had been established beneath the filter-bed, could be leached with great speed under a pressure of 30 to 50 lb. per sq. in." Had this result been attained by applying hydraulic instead of pneumatic pressure, one could understand it ; but how such an extraordinary difference could be brought about by substituting artificial air pressure for natural air pressure is not explained, interesting as it would be to students of physical laws. The method is said to have been applied at a works in Denver and at two in the Black Hills. With these latter, the author is familiar. One was shut down after a very

* 'Metallurgy of Gold,' pp. 284–5.

short life, and the other used barrel leaching as long ago as 1892.

Tank leaching under pressure must be much more costly in plant and in working than free leaching, and, where the nature of the ore demands some such aid, it is a question whether equally good results at less cost may not be got by multiplying the tanks and diminishing the thickness of pulp to be leached.

Leaching in centrifugals was a feature of an Australian patent by Sutton some years ago, and was successfully tried in Queensland. In 1899, the author encountered the co-inventor of Sutton's process, who for some years past has been managing the chlorination and cyanide plant at the Long Tunnel mill, Walhalla, Vic., and was assured by him that the centrifugal leaching was most rapid and satisfactory so long as too great quantities were not treated at a time ; and that the power for causing rapid rotation of the basket was practically nil, and could not be compared at all with that necessary for producing direct pressure or a vacuum.

Washing in the filter-press again is a solution of the difficulty which has been established in cyaniding slimes, a final air current effectually driving out the residual liquor (3 to 6 %) which would otherwise be thrown away with the solid cakes of tailings.

Leaching in the barrel necessitates considerable complication of the latter by adding to it a filter or "diaphragm," forming the chord of an arc of the circle of the barrel. At the Black Hills mills, the filtering medium is asbestos cloth, woven much like gunny sacking, but of stouter thread, supported by a framework, which in some cases is an iron grating coated with lead (a most tiresome and unsatisfactory proceeding), and in others consists of oak planking (which answers well). The 11 × 2-in. planks are carried by wooden cross-pieces resting on bars longitudinally bolted to the barrel, and scored with numerous little grooves centering in perforations at short intervals. By these escapes the liquor which passes through the asbestos cloth lying on the planking, and the cloth is secured in place and at the same time partially pro-

tected from the scouring of the ore, by a movable wooden grating wedged in position. These cloths are a source of considerable expense and delay in operation ; costly in the first place, and enduring only for 2 to 3 days, their renewal occupies 1 to 2 hours.

Elsewhere, for the oak frame is substituted ½-in. sheet lead with ⅜-in. perforations, asbestos cloth being used as before ; or again, the filter is made of 4-lb. lead abundantly provided with fine punctures, supported by a sheet of ⅜-in. lead with ⅜-in. holes, and firmly held by an upper and a lower grating of hard wood keyed in position.

But however arranged, the wear and tear are excessive, and renewals are incessant.

Hydraulic pressure is used in leaching, and commonly ranges from 20 to 50 lb. per sq. in. This is often attained by placing the water-supply tank at a suitable height, but sometimes direct pumping is relied on, and in a few cases both alternatives are available.

Before leaching commences, the barrel is stopped with the charging doors uppermost, so that the filter will be level. Connection is made between the liquor outlet beneath the filter and the settling-tanks. Clean water under pressure is admitted on top of the charge, and, absorbing all free chlorine remaining in the barrel, forces the solution through. The total time employed in leaching out the strong gold liquor, and passing the necessary wash-waters through to remove all traces of gold, varies from about ¾ hour to 4 hours, according to the pressure used and the leaching qualities of the ore, aluminous and slimy materials demanding the maximun time.

Sometimes the leaching and extraction are sought to be improved by interrupting filtration at intervals, and rotating the barrel with plenty of fresh water, so as to avoid formation of prevalent channels of escape for the liquor through the solids.

The volume of fresh water needed is generally 1½ to 2 times as great as that consumed in chlorinating, and is asserted by Thies to be double as much as is required in tank leaching,

while the time occupied is longer and the extraction is less perfect.

Frequently the last wash-waters, being very poor in gold, are not mixed with the strong liquor, but are elevated to a separate tank and used in subsequent leachings.

When the (tested) effluent liquor or wash ceases to give a reaction for gold (p. 431), the barrel is half-turned, to bring the charging-doors underneath, and the contents are dumped into a truck or a launder for wheeling or sluicing away as tailings.

Another half-turn is then given, and the filter or diaphragm is washed by a small hose under pressure ; after a revolution or two, the door is opened for a final discharge of rinsing water, and the barrel is ready for another charge.

Sometimes a long launder is used for conveying the tailings away, and has blanket or coir-matting laid in it to arrest any particles of coarse gold that may have resisted chlorination ; riffles are occasionally provided for a similar purpose.

**Treatment of the Gold Solutions.** *Settling.* — The liquors carrying gold in solution always contain some impurities, such as chlorides of base metals, &c., in solution, and suspended matters if anything goes amiss with the filters. It is most desirable that these shall be eliminated before they reach the precipitation tanks, or the volume of auriferous mud thrown down will be much increased, and the recovery of clean gold bullion made more difficult.

The solutions are therefore run into a series of capacious settling-tanks, built of wood and lined with 6-lb. lead. Here they are treated with strong sulphuric acid, usually about 3 to 6 lb. acid at 66° B. per oz. gold estimated to be present. This is advantageous in two ways. Firstly, gold precipitates from its chloride solution much more freely and fully in presence of sulphuric acid, so that the deposition is hastened and perfected. Secondly, the sulphuric acid throws down as insoluble sulphates such bases as may have come through in the form of soluble chlorides. Most prominent are lime, magnesia and

lead, each of which is almost always encountered, and, even when in very small proportion, is highly objectionable in the gold precipitate. A common arrangement of the tanks is to let them overflow from one to the next throughout a series, the quite clear liquor from the last being pumped up to the precipitation tanks, situated at such a height (generally about 25 ft.) above the filter-presses as to secure the necessary pressure for forcing it through them. In the first settling-vat, to which the acid is added, the calcium sulphate crystallises out, forming a crust on the sides, and, at the bottom, a mud contaminated with the other matter thrown down.

This treatment seems preferable to that suggested by N. A. Ferry,* of adding a dilute solution of molasses (1 gal. in 30 to 40 gal. water) to the unsettled liquor in the precipitation tank, whereby deposition of the lime is prevented. By Ferry's proposal, settling-tanks would be done away with, but no important works have been erected without them.

*Precipitation Vessels.*—The form of the vessel in which precipitation of the gold is carried out varies according to the

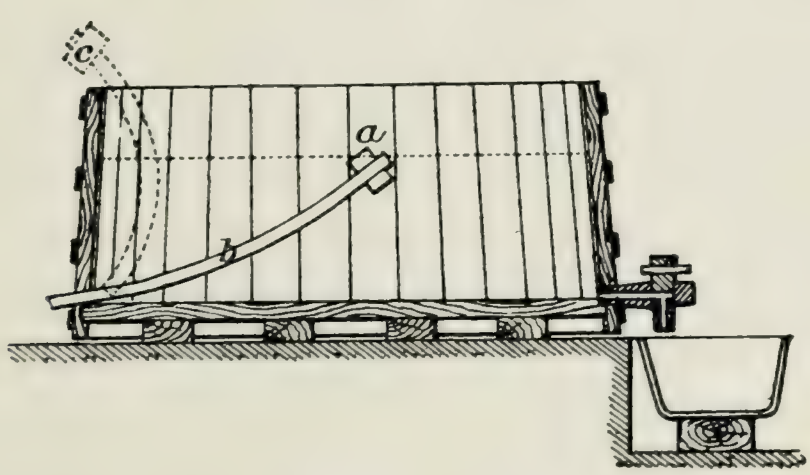



FIG. 134.—PRECIPITATION TANK.

precipitant used. A common pattern adopted when ferrous sulphate or sulphuretted hydrogen is employed is shown in Fig. 134. It is simply a wooden vat, measuring 5½ to 6½ ft. diam. at top, and 6½ to 7 ft. at bottom, and having a depth of 2 to 3 ft., with no false bottom. It is made larger at bottom than at top to prevent the adherence of gold to its sides, and is coated with some preservative such as tar or paraffin paint.

* En. & Min. Jl., Nov. 28, 1885.

The sides are made as smooth as possible. The bottom has a slight dip towards the outlet plug, and must be absolutely free from joint, crevice, or inequality in which gold can lodge ; it is often faced with Portland cement, or a mixture of cement and asphalt heated together and dressed while cooling. A lid which can be secured under lock and key is provided.

Precipitation tanks are kept covered and locked, except when it is necessary to have them opened to remove the gold or to clean them. They absorb a certain amount of the solution, and when old or leaky, are burned to recover the gold from the ashes. They, however, last a very long time.

Sometimes porcelain or glazed-earthenware pans of about 30 gal. capacity are preferred, arranged in a special room ; they have the advantage of being entirely non-absorbent, and are capable of being absolutely emptied by means of a rubber squeegee without an excess of water for sluicing out, as wooden tanks often require.

Other kinds of precipitation vessels will be described under the various precipitants.

*Precipitants.* Ferrous Sulphate.—Of these, perhaps the most generally used is ferrous sulphate ($FeSO_4$), which can be prepared at the works. It is made by pouring 1 gal. sulphuric acid into a tub 4 ft. diam. and 2½ ft. high, filled with scrap iron covered with water, so that there is always an excess of iron ; 1 gal. acid makes a tubful of solution of the requisite strength (about 7 % acid). This solution is siphoned off into a vat 2½ ft. diam. at top, 3 ft. at bottom, and 3 ft. high, and is drawn thence to be used in the precipitation tanks by a rubber hose, which is hung up when not in use. When green vitriol (iron sulphate) can be had cheap, a saturated solution of it is made, and filtered before being used ; but generally iron and sulphuric acid are cheaper.

Sometimes the gold liquor is admitted to the tank first and the ferrous sulphate is added to it, while at other times the procedure is reversed. In either event, it is well not to have the precipitant in excess, but rather to make successive further additions of it till all the gold is thrown down, and to let thorough agitation accompany each addition.

A long period is required for settling—usually not less than 48 hours, and sometimes 60 hours is not sufficient. So long as a faint purple tinge can be seen in the liquor, there is gold in solution. It is probably never entirely deposited. The time usually depends on the heat of the weather, taking much longer in winter than in summer.

When the clear supernatant liquor shows no gold on testing, the upper part of the liquor is removed with a siphon, or by tapping the tank at different levels, as little liquid as possible being left on the precipitate. As both these methods require watching, the plan generally adopted is to use a flat piece of wood *a*, attached to a rubber hose *b*, Fig. 134. This is done in such a way as to keep the mouth of the pipe always under the level of the water, until there would be danger of the gold on the bottom being disturbed, when the width of the float prevents the liquor from being drawn off any further. When not in use, the pipe and float are hung up, as at *c*. This method requires no watching.

All the liquor flowing off after precipitation of the gold is collected in large tanks, and allowed to settle until place is required for other liquor, when it is drawn off. Some little gold is caught in this tank, but it is only cleaned up at long intervals, generally once a year.

As it is never certain, even when liquor appears quite clear to the eye, that no fine gold is in suspension, it is always safer to allow this liquor to flow through a filter of sawdust before running to waste; this sawdust is burned after a time, and the gold is recovered from the ashes.

The precipitated gold, unless the ores are very rich, is allowed to collect for several operations. It is then dipped off from the bottom of the tank with a clean porcelain or enamelled dish, and the bottom and sides of the vat are thoroughly brushed with a small brush, and washed with the least possible quantity of water, the final residue being sluiced out with the hose from the bottom stop-cock of the vat, settled, and filtered.

This method of precipitation is difficult, because the fine gold remains so long in suspension, a purple colour being

seen in the liquor for several days, and because all the wash-water must be kept, on account of the small quantity of gold it is liable to contain.

The precipitate obtained is treated with a mixture of sulphuric acid and salt to free it from iron. It is first red, and changes afterwards to a brown-gold colour.

The purified precipitate is placed in a cloth and strained, and is then squeezed by twisting to make it as dry as possible. When it is moderately pure, it will fall after squeezing with a touch of the finger, but when filled with impurities it will not.

These cloth filters are always kept under water while not in use. When worn out, they are dried, a little nitre is sprinkled over them, they are burned, and the gold is recovered from the ashes.

The gold is sometimes collected on a filter made of punched sheet iron covered with two or three thicknesses of ordinary filter-paper, and simply washed with hot water until it is sweet.

An innovation by Munktell, at Fahlun, Sweden, is to heat the liquor to 160° F. when precipitating by ferrous sulphate, and to add a little plumbic acetate, which, being thrown down as lead sulphate, helps to carry down the gold particles, hastening and ensuring their deposition.

A centrifugal separator, as proposed by Sutton and successfully tried both by him and by Vautin, would seem well adapted to this method of precipitation, as its great drawback is slowness of deposition. From experiments made by the author with gold-zinc muds from cyaniding, a centrifugal with walls of fine wood-charcoal packed tightly between asbestos or other woven material promises complete success.

Hydrogen sulphide.—Sulphuretted hydrogen ($H_2S$) is in general favour in the United States; by it the reactions are complete in an hour, but the gold is liable to contamination by copper, iron, or lead in the ore, unless care is exercised. The gas may be prepared from paraffin and sulphur, or from iron matte and sulphuric acid.

In the former case, the generator is a small cylinder of

heavy boiler iron, with cast-iron heads riveted in, and all the seams well caulked. In one of the cast-iron heads is a hand-hole, used to clean out the residues from the charge ; and in the shell of the cylinder is the charging hole, and a piece of 2-in. gaspipe, the outlet of the gas. The charging hole is fitted with a tight cover, easily removed. The generator is built in a small brick fireplace, leaving the charging hole and gas outlet on top free, and the discharging hole accessible from the outside. After charging with the proper proportions of paraffin and sulphur, a light fire is all that is necessary to generate gas. The best proportions, using single-pressed paraffin and common brimstone, are 1 to 2.

For iron matte and sulphuric acid, the generator is a small iron lead-lined tank, as in Fig. 135. When charged with matte and dilute acid, air is admitted under pressure at *a*, and the gas is forced through a hose or pipe into the precipitator. The iron sulphide rests on a perforated leaden shelf *b*, fitted with a pipe *c*. Acid admitted at *d* flows to the well *e* and is forced by the air up the tube *c*.

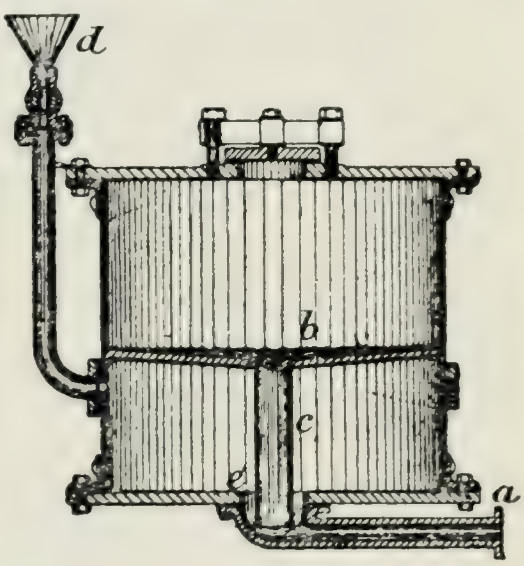


FIG. 135.—HYDROGEN SULPHIDE GENERATOR.

The precipitation tanks are fitted with a tight cover, having an exhaust-pipe for the escape of gas during precipitation, and with a manhole having a tight cover. Near the bottom of the tank is a perforated lead pipe, one end coming up through the cover and fitted with a hose connection. In front of the precipitation tanks is a line of steam and gas pipes, fitted with T's, valves, and hose connections for each tank.

The gas from the generator passes first into a receiving tank, where the volatile oils, etc., that will condense, can collect.

From the receiving tank, the gas is drawn by a small air-pump, and forced through the line-pipes to the tanks, and through the solution to be precipitated. To avoid creating a

vacuum in the generator and gas receiver, a small air-hole is left in the receiver. In this way, the pump is able to work much faster, and, by driving air through the solution, materially aids the precipitation, and reduces the amount of precipitant used.

After precipitation is complete, the tank is allowed to stand for an hour, and then several small pressure-filters are attached, and the solution is allowed to flow through them ; 3 of these filters will empty a 1500-gal. tank in 3 to 4 hours, and leave it ready for another precipitation ; or the precipitate can be collected in another filter-press.

The precipitation tanks are carefully swept on clean-up days (fortnightly or monthly), and all matters collected are passed through the filter-press.

Filter-presses are now made in a great variety of patterns, but all consist essentially of a series of round or square plates, forming filter chambers, either by their concave surfaces, or by means of frames interposed between them. These plates and frames are inserted between a head and a tail block, and are held in place by tie-rods or side-bars. A central screw or hydraulic piston forces the head-block, plates and frames together. Filtering cloths of suitable material are placed between the plates and frames, forming tight joints on the edges. The material to be filtered is then forced into the series of chambers by a pump. The liquid passing through the filtering cloth is conducted away through proper channels and outlets ; the solid material remains in the chambers in the form of a cake. The firmness and dryness of this depends upon the nature of the material, and the pressure used in forcing the liquid through the chambers. When the chambers are full, the press is opened and the solid residues are removed.

English makers have always been prominent in this department, and a Johnson press is shown in Fig. 136. Canton flannel is used as a filtering medium, some 54 yd. of the quality known as "A 2, 8 oz." being required to make double cloths for a press containing 24 plates, 18 × 19 × 1 in. Pressure forces the liquid into the apertures shown at *a*, whence it flows to taps *b* and into the trough *c*.

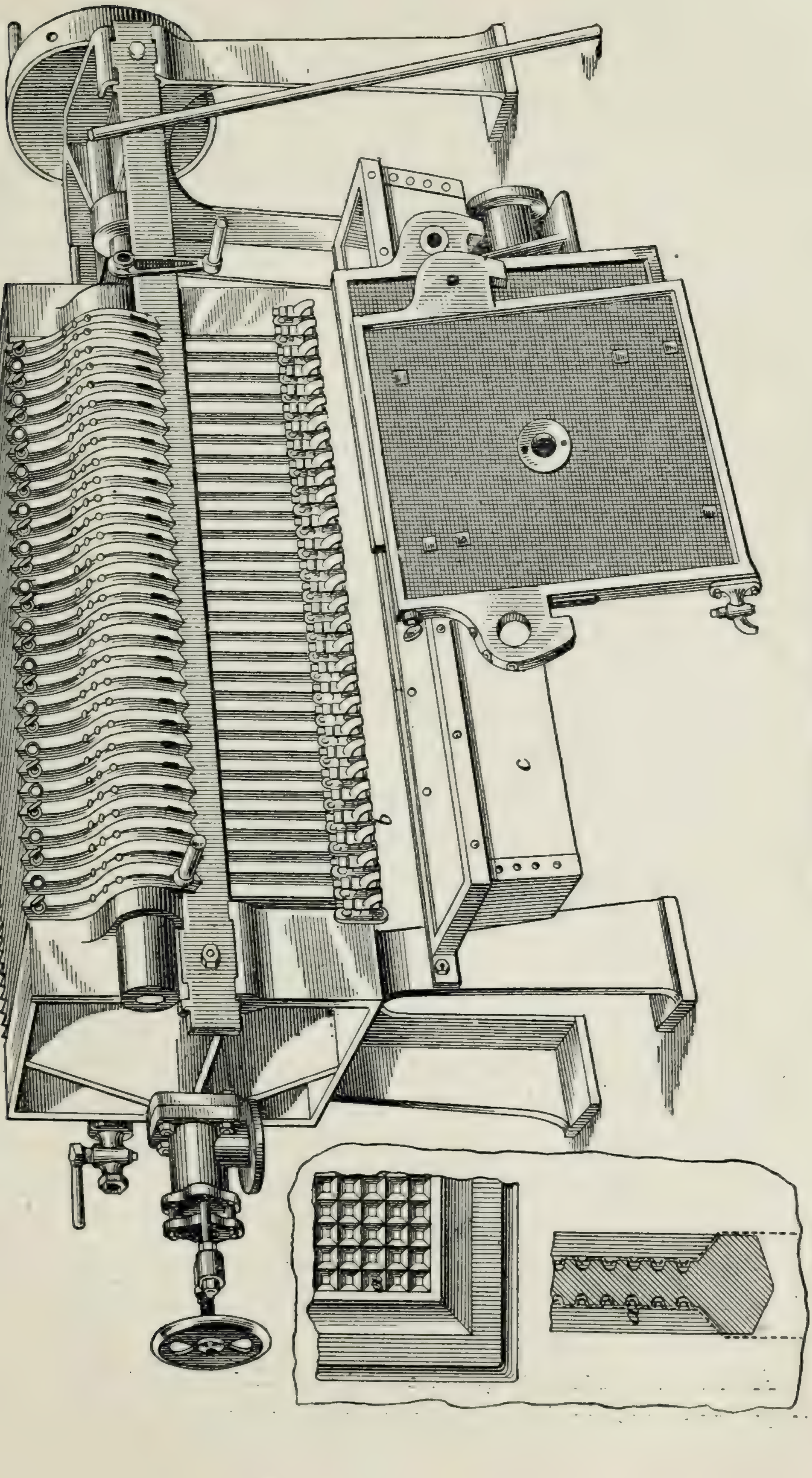



Fig. 136.—Johnson Filter-press.

Sometimes paper sheets are used to catch the solids, and are afterwards burned in special receptacles to prevent loss by dusting and draught.

The passing of sulphurous acid ($SO_2$) into the liquor is a usual preliminary to admitting the sulphuretted hydrogen, in order that all the chlorine may be converted into hydrochloric acid, and precipitation of sulphur from the $H_2S$ be avoided. (See p. 443.)

Wood Charcoal.—Wood charcoal is a favourite precipitant in Australia, being more used than any other. At the South German mill, Maldon, Vic., the charcoal is crushed in a sort of magnified coffee-mill to about the size of horse-beans, and is levigated to remove fine dust. The coarse material is packed into small filters of very simple construction, being merely glazed drain-pipes, 8 in. diam. and about 2½ ft. long, standing in wooden tubs, 12 in. diam., and stayed. The charcoal is filled into both the drain-pipe and the space around it, and is kept from being carried away by the current by a covering of wire gauze and a few large flints. The stream of liquor is split up between 6 filters in a row, flowing first down the drain-pipe, then ascending by the outer packing, and escaping by a short tube to a second series, in the same way. When the author visited the works, in 1899, there were 6 series in operation, and their efficiency was such that the escaping liquors did not assay 1 gr. per ton. Each row or series stands on slightly lower ground, to enable the liquor to gravitate through them. It was found that the great bulk of the gold was deposited in the first row. Such weak chlorine waters are used for leaching that no chlorine remains for "killing," and the gold liquors pass direct to the filters without being heated.

A very similar installation has been in use for years at Bendigo, the charcoal being packed in earthenware pots about 4 ft. deep; and much the same procedure is followed at the Long Tunnel, Walhalla, and at many Ballarat mills, ½-in. charcoal being popular.

At Mount Morgan, Q., the largest chlorination mill in the world, the charcoal is crushed and sifted; all that passes

20-mesh and remains on 30-mesh is called "coarse;" that which passes 30 and remains on 40 is called "fine;" all that passes 40 is thrown out; the "fine" and "coarse" are washed with water to free them from dust. The filters can be any size desired, but it is well not to have the charcoal column less than 18 in. in depth. The small filters contain 12 cub. ft. of charcoal, measured loose, i.e. 4 cub. ft. of "coarse" and 8 cub. ft. of "fine," 1 cub. ft. of "coarse" being placed on the bottom, and the 8 cub. ft. of "fine" above this; on top is put the remainder of the "coarse," with a heavy perforated sheet of lead to keep it down. The charcoal is stamped in hard with the foot, and especially around the circumference of the filter, to prevent the gold liquor finding its way between the charcoal and the walls. The liquors running away from these filters assay ·68 to 1·69 gr. per ton. After passing the charcoal, all liquors are run through a concrete tank with a bed of sawdust 1 ft. deep, and covered with lead plates like the filters. Every 6 months this is burned, and 15 to 20 cwt. of ferric oxide ash is obtained, assaying 200 to 300 oz. gold per ton, which is sold to smelters. The filters are now made very large, and lined with cement.* In 1887, when strong chlorine solutions were used, the gold liquor was boiled to expel chlorine before going to the charcoal filters; now, with very weak chlorine waters, it is only warmed.

The impregnated charcoal, freely coated with gold, is carefully collected from the filters, and dried and calcined on iron trays, with the least possible draught, in a furnace which can be locked up. At Maldon, this is a small brick muffle, and the arrangements are not particularly well designed for preventing loss from dusting. At Walhalla, a Turnbull furnace with special provision for avoiding gold loss is employed, and is very highly spoken of. The burning off is a somewhat tedious process, and, if unduly hastened, may result in extravagant loss of gold; but that seems to be its only drawback. Various species of gum-tree (eucalyptus) are used in making the charcoal, the harder-wooded varieties, such as "box" and "stringy bark," being preferred.

* E. Hall, En. & Min. Jl., Oct. 7, 1899.

The calcined residue should be washed with hydrochloric acid and with warm water, to remove soluble salts before melting, or the destruction of graphite pots may be excessive.

Copper sulphides.—Copper sulphides have long been before the public as precipitants. C. H. Aaron recommended the precipitated sulphide ($CuS$) in 1888; but in practice its physical character and proneness to oxidation, forming $CuSO_4$, debar it. In 1891, Blomfield expressed a preference for the sub-sulphide ($Cu_2S$), formed by fusing together sulphur and copper, and found it cheaper. The "white metal" of the copper smelter, when free from silver, is equally good. Either is crushed to pass through 60 holes to the linear inch, and the portion that remains on a 100-hole sieve is used. The reason for excluding the very finely-divided portion (that passes through the 100) is to facilitate percolation of the gold solution.

The most convenient manner of using the reagent is to arrange a shallow bed, and allow the solution to run through. It is preferable to use three vessels, so arranged that the liquor falling from the first may run into the second, and from the second into the third. The first of the series can be removed when necessary; the second is moved up to take its place; and fresh material is placed in a third. The first vessel should not be removed until gold is clearly seen to be deposited in the second, so as to ensure the almost entire decomposition and removal of the precipitant from it.

"If this part of the operation is watched, all that is requisite will be to dry the contents of the vessel before fluxing and smelting to bullion. A great advantage of this reagent in practical work is that it is not necessary to drive out excess of free chlorine from the gold solution; nor does the presence of free hydrochloric acid interfere with the reaction Any copper, calcium, zinc, etc., that may be in the solution also passes through, and here it will be clearly seen that this method presents a great advantage over the use of hydrogen sulphide or ferrous sulphate." *

However, where it has been tried on a working scale it has been found "impossible to displace the whole of the copper"

* C. Vautin, Trans. Inst. M. & M., i. 273 (1893).

in the first vessel, and "extremely difficult to extract all the precious metal from the copper regulus which forms on smelting."* Besides, as Prof. Huntington points out, in presence of free chlorine, other reactions must occur, resulting in cupric sulphate being formed, though this might be removed by washing with nitric acid. Despite its attractiveness and apparent simplicity, the process is not at work in any mill, having been abandoned in favour of ferrous sulphate at the Ouro Preto mill, Brazil, where alone it was adopted.

Other Reagents.—Various other reagents have been proposed, and found advocates, at different times, but none has survived an experimental stage. Among the more important are iron sulphide (FeS), sulphurous acid ($SO_2$), iron filings, and mercury.

**Treatment of the Gold Precipitate.**—When the reagent used affords a gold precipitate in a muddy form, either as a sulphide ($Au_2S_3$) from $H_2S$, or as free gold contaminated with various sulphates from $FeSO_4$, the tank is not disturbed for 2 or 3 hours after all reaction has apparently ceased. Then the supernatant liquor is siphoned off to within a few inches of the deposit, and is passed through a filter-press (p. 454), so as to make sure that nothing of value is carried away in suspension. The precipitate left in the tank is only dealt with at the usual 2-weekly or 4-weekly clean-up, when it also is passed into the filter-press, and all liquid is removed as far as possible in that way. Finally compressed air is forced through the press to dry and harden the cakes, so that they may be more conveniently handled.

The filter-cloths, when they have become clogged or acid-eaten, and unfit for further use, are dried and incinerated for recovery of entrapped sediment.

The gold mud or cake, generally rendered bulky by sulphur and arsenic, and by sulphides of antimony, copper, silver, etc., when using $H_2S$, and by various sulphates when using $FeSO_4$, is very slowly and carefully roasted in iron pans in a brick muffle, or in a cast-iron muffle furnace, heated from

* S. J. McCormick, Trans. Inst. M. & M., v. 120 (1897).

above, and requiring neither stirring nor passage of a draught through the furnace, so as to avoid loss by dusting. After roasting, when the mass contains, as it usually does, oxides and basic salts of iron, etc., it may advantageously be treated with hydrochloric acid and with warm water to dissolve them out as far as possible. This done, the relatively clean bullion is mixed with the necessary fluxes, such as nitre and borax, and sometimes salt or carbonate of soda, and melted in clay or graphite pots. The slag resulting from the melting is crushed and melted down with litharge, and the lead is cupelled.

The clean-up from charcoal precipitation is much more simple, the charcoal being burnt off as described, and the resulting ash being fluxed and melted after removal of soluble salts (p. 458). At Mount Morgan, when barrel chlorination was practised, it was customary to amalgamate the ash from the supplementary charcoal filters, as it carried only 5 to 10 % gold; but since vat lixiviation has been established, the charcoal ash assays 30 % gold. This, by amalgamation, produced very rich slimes, worth 1200 oz. gold per ton, which were sold to smelters; but the loss by this method was high, and moreover it is undesirable to put so much value into the form of a by-product. Smelting the ash in crucibles was then adopted, but the amount to be smelted was about 30 cwt. (60 cwt. with fluxes) per month, making the operation very laborious, while the slags were always full of gold beads, which had to be amalgamated. A small reverberatory furnace was then designed, with a cast-iron pan to hold the brick hearth. The pan is supported on four screw standards, which can be moved up and down, and the whole hearth may be withdrawn from the furnace when necessary. Smelting in a sand hearth was tried, but was given up in favour of this simple fire-brick hearth.

The subject is further dealt with in Chapter XII.

**Working Results.**—In comparing the working results obtained at different mills in various parts of the world, the greatest divergence is noted, both in extraction and cost. For

the most part, the figures bear some relation to each other, improved extraction resulting in increased cost; but absence of uniformity in the ores dealt with is largely responsible for apparent contradictions.

Owing to limited production of concentrates at most gold mills, not being in sufficient quantity to afford constant supplies for the steady running of a chlorination plant, this business is very largely done by custom mills, which buy the output of the neighbouring batteries. Competition has something to do with the prices and conditions offered by these establishments. The most common arrangement is to receive the concentrates in sacks (see p. 359); each sack is weighed and sampled, the samples being immediately sealed up in bottles to prevent loss of moisture, which has to be estimated and deducted from the nett weight of ore delivered.

*Extraction.*—On the average, payment is made for 90 % of the fine gold determined by assay at this stage; but sometimes the buyer roasts before assaying, to guard himself against possible gold loss in roasting (see p. 367). In some cases 85 % only is returned; while in others, an ascending scale is adopted, according to the richness of the material, thus 90 % on 2½-oz., 92 % on 5-oz., 93 % on 10-oz., 94 % on 15-oz., 95 % on 20-oz., and 96 % on 25-oz. Silver contents are disregarded, unless they are so large as to warrant separate treatment, when special terms are made for them. The Golden Reward Co., Black Hills, allows 90 % of the fine gold, but refuses all ore containing more than 3 oz. silver per ton. It is claimed that their extraction ranges between 90 and 98 % on 15 to 16-dwt. ore, but assays of various samples of their tailings gave the author from 4 dwt. to over 7 dwt. per ton (roasted), which will not give a much better average than 75 %.

Prof. Jenney states the figure at 80 to 87 % for the mills of the Black Hills.

The Nevada City chlorination works, California, return about 93 % on ordinary 2- to 3-oz. Grass Valley concentrates.

The Delano mill, Colorado, gets 92 % out of ·65-oz. telluride ore.

The Haile mill, Carolina, recovers 94 % from 2½-oz. concentrates.

At Fahlun, Sweden, Munktell obtained 90·32 % from copper ore tailings assaying only 1·55 dwt., and 98·85 % from ore carrying ·97 oz.

Mount Morgan extracts 94 % from the sulphide ore of 16 dwt., and 96 to 98 % from the oxidised ore.

The South German mill, Maldon, Victoria, claims a 97 % extraction on concentrates.

At Ouro Preto, Brazil, 93 % is returned from 1·635 oz.

The Robinson mill, Transvaal, shows 96 % on 5-oz. concentrates from the Ferreira Co.

*Cost.*—In quoting the following costs for vat and barrel treatment respectively, no account is taken of the item roasting.

At the Treadwell mill, Alaska, in 1897, over 3000 t. of 1½- to 2-oz. concentrates were chlorinated in vats, at a cost of 22*s.* 1*d.* per ton, and in 1898, 7708 t., at a cost of 15*s.* 6*d.* per ton. These works have now been closed down, and the material is shipped and sold to the Tacoma Smelting Co., Washington.

In 1891, Munktell treated some highly refractory concentrates (·768 oz. per ton) at Brad, Hungary, for just under 22*s.* per ton.

Tailings from copper works at Fahlun, Sweden, yielded the same operator over 90 % on 1½ dwt. at the remarkably low cost of 8*d.* per ton.

At Mount Morgan, in 1899, where over 200,000 t. of ore yearly are chlorinated, the oxidised ore cost 3*s.* 1½*d.*, of which chemicals amounted to 11½*d.*, and labour and fuel to 2*s.* 2*d.*; the sulphuretted ore absorbed 3*s.* 10*d.* worth of chemicals.

Operations on concentrates at a Brazilian mill (1899) cost 8*s.* 4*d.* per ton for chemicals, etc., and 5*s.* 10*d.* for labour, but the waste of chlorine is great.

Among establishments using the barrel process, those in the Black Hills are perhaps most important; here the cost for 16-dwt. ore crushed to 8 mesh is about 7*s.* per ton (1891).

At Haile, Carolina, in 1895, on 2½-oz. concentrates, it was

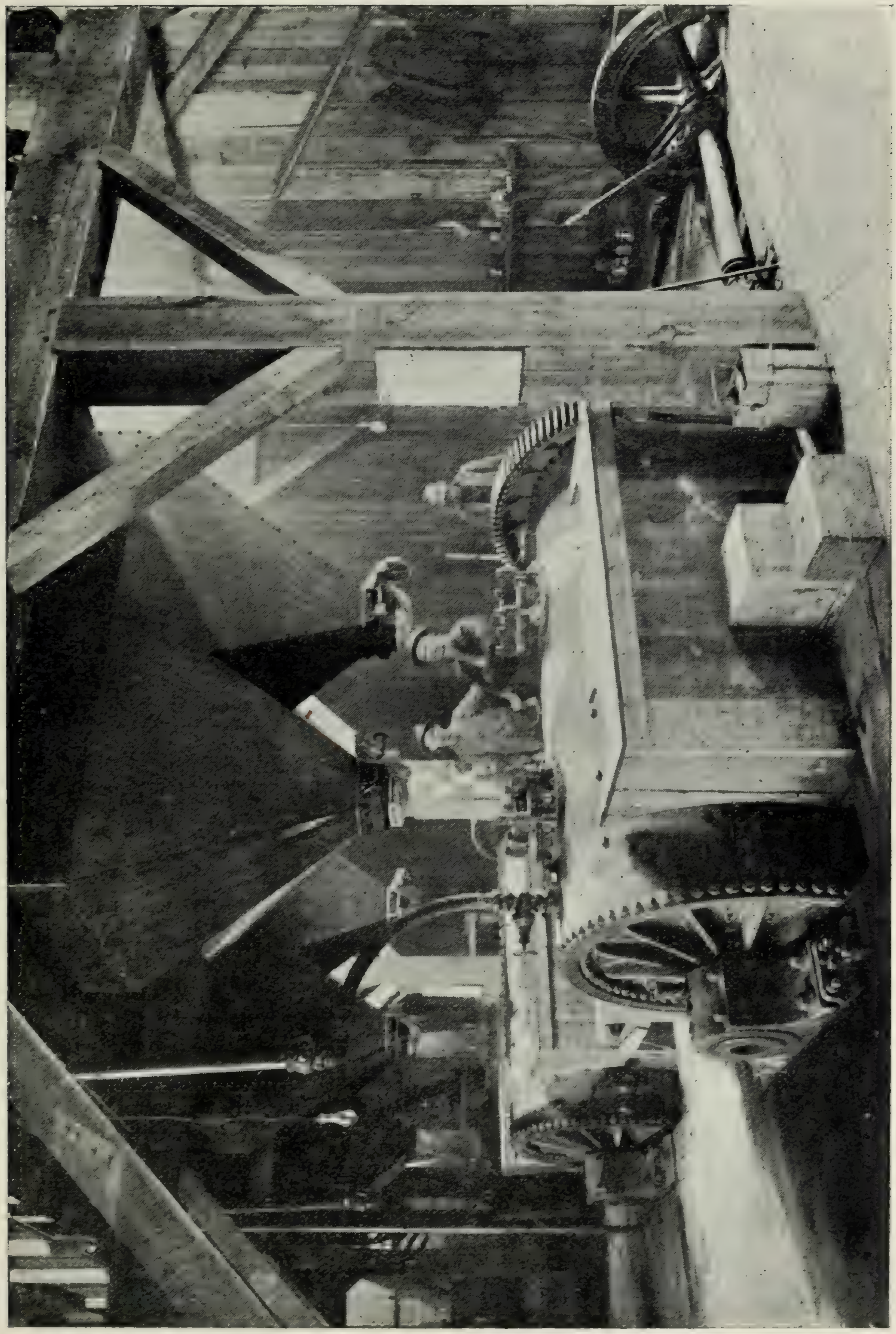


FIG. 137.—CHLORINATION BARREL ROOM.

8*s.* 5½*d.*; and at Delano, Colorado, in 1898, on ·65-oz. telluride ore, 10*s.* 7*d.*

In the Transvaal, 2 oz. per ton is considered about the minimum for concentrates adapted to chlorination; for lesser values, cyanidation (raw) is preferred.

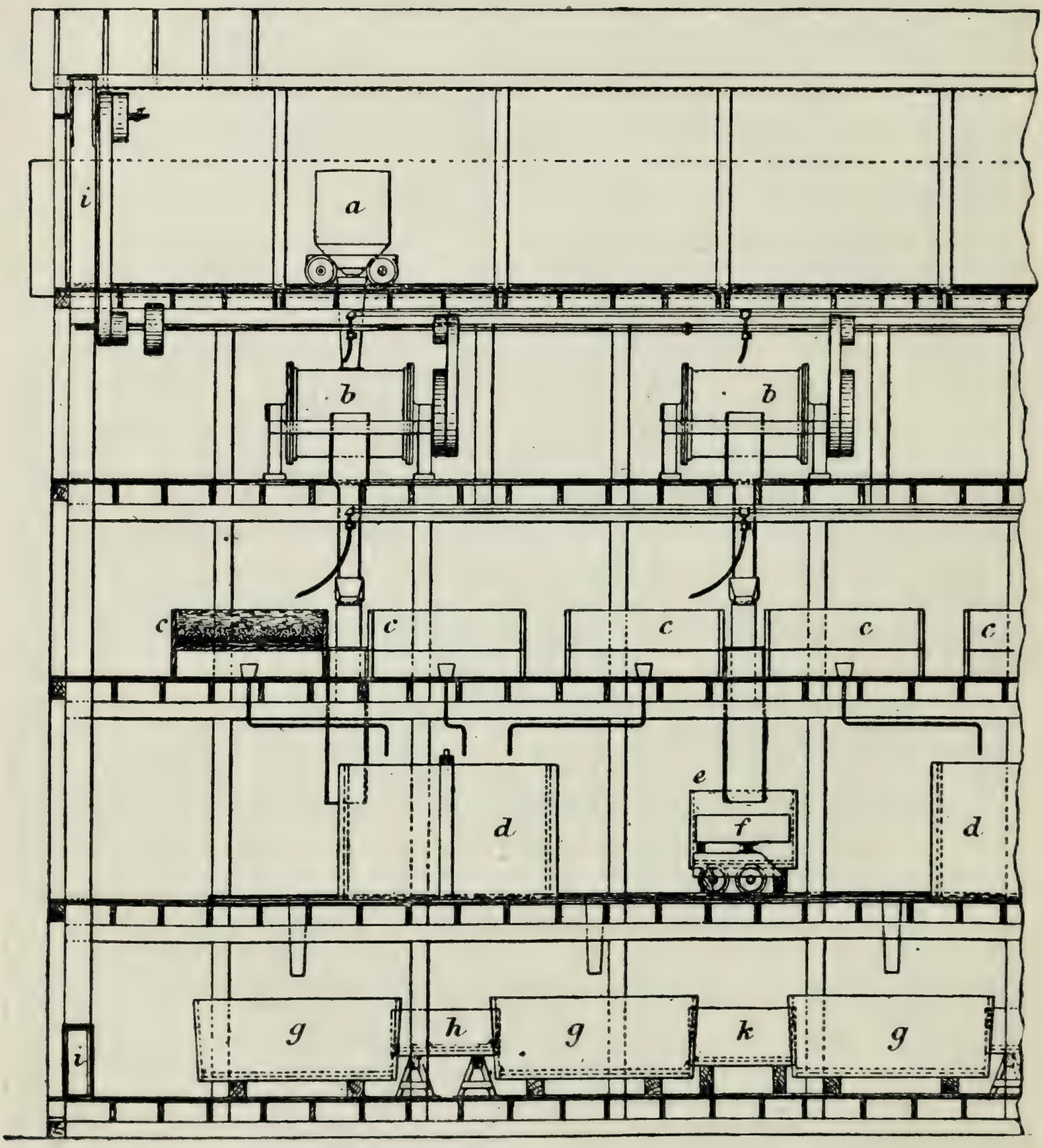



FIG. 138.—CHLORINATION MILL.

The public mills of Colorado, operating on telluride ores up to 2 oz. gold per ton (richer ores being smelted), charge at the following scale:—up to ½ oz. per ton, 29*s.* 2*d.* per ton; ¾ oz., 32*s.* 3*d.*; 1 oz., 35*s.* 5*d.*; 1¼ oz., 37*s.* 6*d.*; 1½ oz., 41*s.* 8*d.*; 2 oz., 43*s.* 9*d.*

*Plant.*—A general view of a barrel-room is shown in Fig. 137, representing an installation by the Gates Iron Works, at one of the new mills in the Black Hills, S. Dakota.

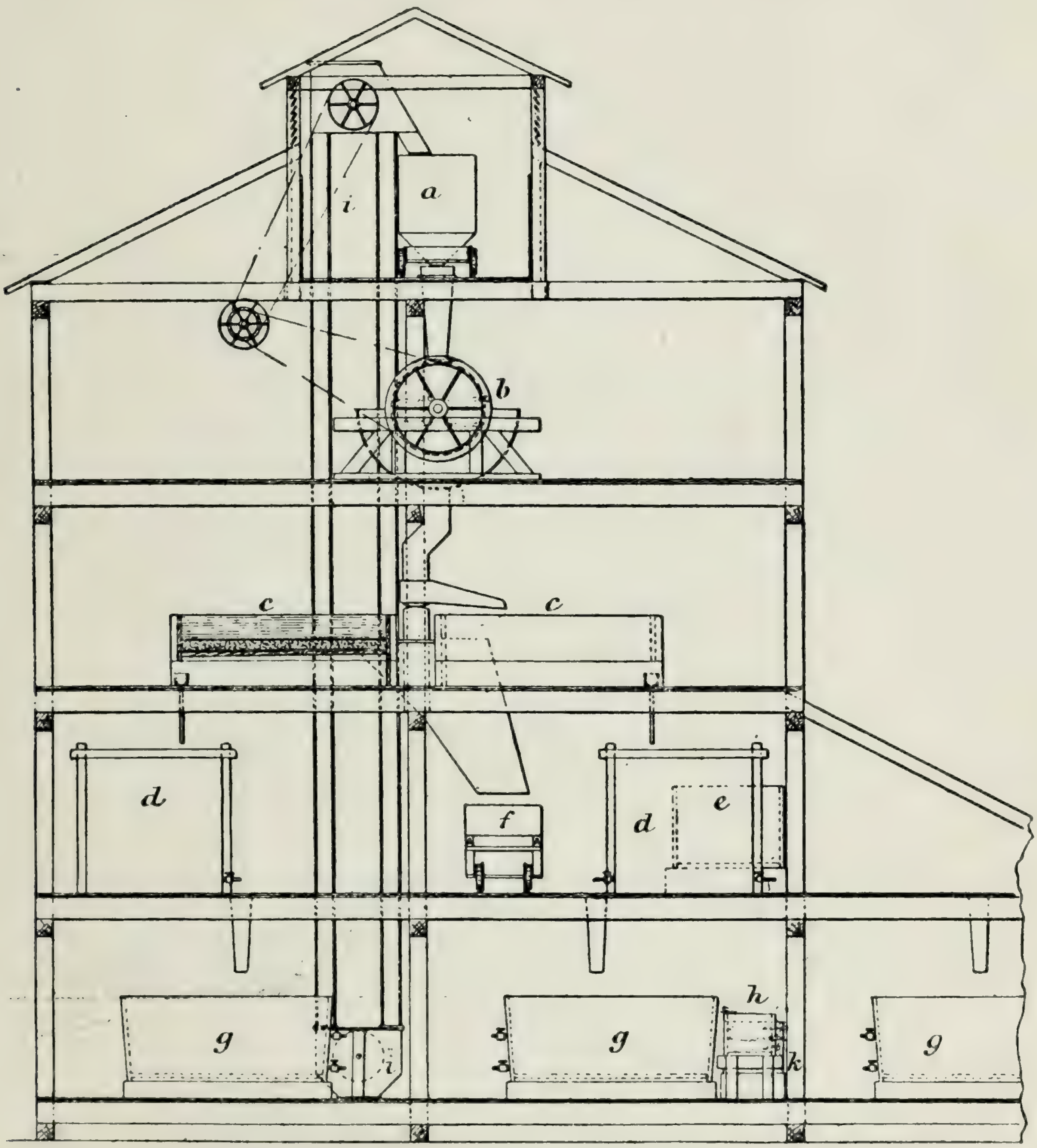



FIG. 139.—CHLORINATION MILL.

Figs. 138 and 139 illustrate a complete chlorination plant, excluding the crushing and roasting appliances, as used in the Thies method, when the leaching is done outside the barrel. The car *a*, running on a light track, conveys the crushed and roasted ore to the series of barrels *b*. These discharge into the leaching-tanks *c*. The liquors, escaping by

the funnel-pipes shown, pass to the storage tanks *d*, and are thence drawn off as required to the precipitation tanks *g*. A supply of sulphuric acid is kept in the tank *e*. Solid residues from the leaching-tanks are discharged into hoppers leading to the car *f*, for trucking out of the works. *h* is a settling-tank ; *i*, an elevator for raising ore to the barrel-floor ; *k*, store-tank for ferrous sulphate precipitant. For a flat site, the arrangement is good, as it permits the spent ore to be dis-

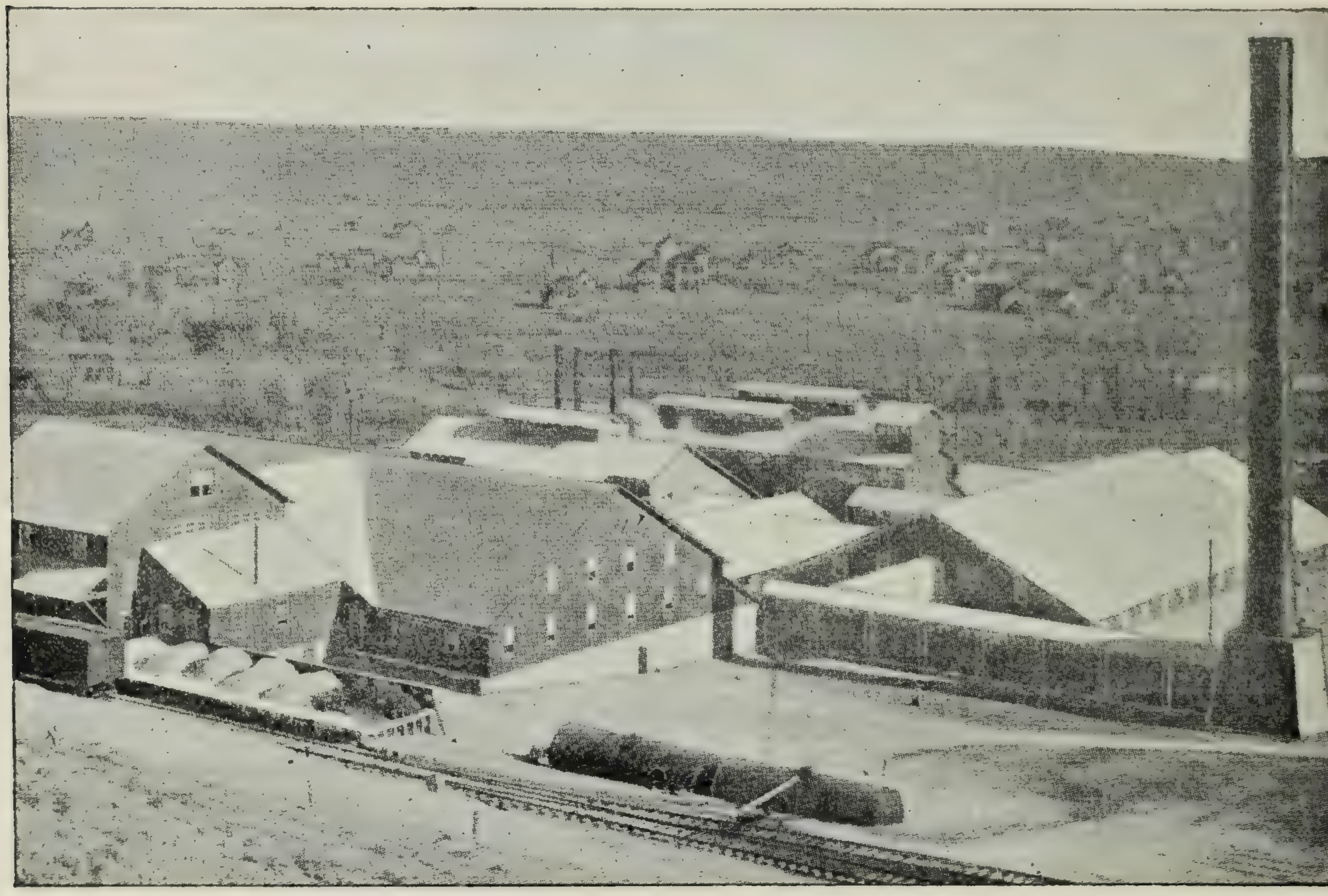


FIG. 140.—CHLORINATION WORKS.

charged at a high level without incurring a second lift. But where there is a natural fall and convenient get-away for solid residues, it would seem to be certainly one if not two stories higher than is necessary, involving much additional cost for erection of buildings and for elevating ore ; and even though the introduction of pumps may be required to overcome the absence of fall, that is a common practice in other mills, and is not found to be a serious drawback.

At the Colorado City works, Fig. 140, built by the E. P.

Allis Co., barrel leaching is adopted. The chlorination department occupies a main building 180 ft. long and 28 ft. wide, with an annex for sand-filters 150 by 22 ft. The main building is equipped with 10 barrels, each 6 ft. diam. and 12 ft. long, piped so that pressure either from the main water-supply tank or direct from a steam-pump located in the barrel-room can be used ; driving gear is all under the floor, so that there are no overhead belts. Over each barrel is a charging hopper, having a capacity of one charge of ore. At one end of the barrel-room, on the same floor, is the chemical store-room, whence acid and bleaching powder are drawn to the barrels in buckets suspended from overhead crawls. The settling and precipitating department is in a building 97 × 45 ft. On the second floor are the settling-tanks, having a capacity of 80,000 gal.; on the first floor, the precipitation tanks, of 60,000 gal. ; and on the ground floor, the montejus or pumps, pressure-tanks, and filter-presses, as also, in a separate room, the muffle furnace for roasting precipitate, and melting furnaces for the bullion. The power plant is in a distinct building 92 × 42 ft., and embraces 4 100-h.p. boilers, a 150-h.p. engine, an air compressor, and a high-speed engine and dynamo for electric lighting. The works commenced operations in Dec. 1896, and treat 4500 to 5600 t. monthly.* The illustration embraces railway track, with heaps of ore unloaded from cars, as well as the dry-milling and roasting plant already described on pp. 294–5, 393–4.

Some further mills are illustrated in Chapter XIII.

**Bromination.**—There have been many and various suggestions from time to time for employing bromine as a substitute for chlorine. Alone it dissolves none too readily in cold water, and in that condition is by no means an active solvent of gold ; but its energy is increased by adding a little potassium or sodium bromide or even common salt, and by acidifying with hydrochloric or sulphuric acid.

A method was worked out by E. C. Engelhardt, Denver, in 1894, for treating the Potsdam ores of the Black Hills, in

* H. V. Croll, En. & Min. Jl., Oct. 8, 1898.

which he employs a solution of bromine in water acidulated with hydrochloric acid, and adds to the charge in the barrel sufficient soda carbonate to nearly, but not quite, neutralise the hydrochloric acid, unless much lime or magnesia is present in the ore. The bromine would decompose the soda carbonate, displacing the carbonic acid and combining with the soda; but the hydrochloric acid present, being free, attacks the soda, forming sodium chloride, before the bromine can exert any action upon it (the soda). Some experimental tests made for the author gave the following results:—

Size 30 to 80 mesh—

| | | | | |
|---|---|---|---|---|
| Before treatment . . . | 9 dwt. | 7·44 dwt. | 11·8 dwt. | 12·91 dwt. |
| After ,, . . . | 2·1 ,, | 1·44 ,, | 1·6 ,, | 2·06 ,, |
| Extraction . . . . . | 76·6 % | 80·6 % | 86·5 % | 84·0 % |

Size finer than 80 mesh—

| | | | | |
|---|---|---|---|---|
| Before treatment . . . | 9·2 dwt. | 15·08 dwt. | 12·5 dwt. | 13·02 dwt. |
| After ,, . . . | 1·8 ,, | 2·06 ,, | 1·4 ,, | ·93 ,, |
| Extraction . . . . . | 80·4 % | 86·3 % | 88·8 % | 92·8 % |

Trials on unroasted ore gave extractions ranging from 64·4 down to 16·9 %. The presence of tellurium, not then suspected but since discovered, may partially account for such total failure.

In no case do the figures compare favourably with the work done by chlorine. As to cost, there is no basis on which to found a comparison. There would be a material saving in freight on bromine as against bleaching powder with only say 30 % efficiency, but the relative consumption is not ascertained. Further, the price and supply of bromine are controlled by the producers. Moreover, it would seem from some experiments that the presence of much silver would be detrimental to vat treatment, by coating the gold with bromo-silver salts.

A bromination mill erected at Rapid City, in the Black Hills, used 1½ to 3 lb. bromine, dissolved in hot water, per ton of ore, and applied excessive pressure. The extraction obtained was only 60 to 75 %, and the mill soon shut down.

In the Cassel-Hinman process, the ore is treated in open vats by nascent bromine generated among it from a mixture

of bromide and bromate of soda or lime. Considerable excess is used, so that dissolution of the gold may be rapidly effected, while it is expected that the surplus bromine will be recovered for further use. No fumes, it is said, are encountered in working. The consumption of bromine is reported to be about ½ lb. per ton, in addition to which, 3½ lb. caustic soda or lime, 3 lb. sulphuric acid, and "a little chlorine" are needed, the total cost of chemicals being estimated at 1*s*. per ton. Experimental extractions of 95 to 99 % at a cost "estimated to be about the same as cyanide" have been announced.

It is now two years since a heavily capitalised company was floated in London for working this process; but beyond the issue of a most attractive prospectus, little progress seems to have been made. The success of the process is dependent on three principal conditions, each one of which presents many difficulties in practical working.

The first is an impregnation in open tanks of the dry and quite sweet-roasted ore (previously acidified by sulphuric acid) with the bromine solution: both acidification and impregnation must be complete, not merely partial, or there will be gold unattacked and an escape and loss of some of that excess of bromine which is admittedly necessary. With the means proposed, on a working scale, this may safely be said to be impossible of attainment.

The next step is the recovery of the dissolved gold from the auric bromide solution, which is washed out from the ore with water. To this end, as simultaneous recovery of the bromine is aimed at, the gold bromide liquor is passed down a sort of scrubbing-tower against a rising stream of air, chlorine gas, and steam. This is counted on to decompose the auric bromide, chlorine taking its place; but the adjustments necessary to ensure this involve extreme care, and even then there will result complicated reactions and various losses. The conditions presented are very different from the simple features of the Gay Lussac and Glover towers of the sulphuric acid industry, from which the idea is borrowed, and the margin for loss and failure is much narrower, by reason of the higher values at stake. Assuming everything to work satisfactorily,

the gold is still in solution, and only now in the same condition as it would have been supposing chlorine had been used in the first instance, with the possible addition of some undecomposed bromide to complicate the precipitation. This last step is assumed to proceed without hitch of any kind, but experience shows that many difficulties may be encountered.

Finally, the liberated bromine, volatilised from the auric bromide solution in the steam tower, is conveyed into another tower where it meets a falling spray of caustic soda solution, and is converted into a mixture of bromide and hypobromite of soda, being then heated to change the latter into bromate, ready for re-use. What percentage of recovery could be counted on in regular working with such a roundabout procedure, and at what cost, remains problematical in the absence of reliable data from full-scale operations.

It would seem that the Cassel-Hinman, like the crowd of American processes for applying bromine in place of chlorine, must fail in commercial application, both because of the cost of the original reagent and the impracticability of recovering a material proportion of it remuneratively. Whether greater success would attend the use of very weak solutions, operating for extended periods, and making no attempt to recover any portion of the bromine, is yet to be proved. At this moment, no bromination mill, on any system, is actively engaged, so far as the author is aware.

**Recovering Associated Metals.**—Many sulphuretted ores contain silver and copper as well as gold, but chlorination gives no account of them. When their proportion is large, steps must be taken for their recovery, and in such cases recourse is generally had to smelting, instead of chlorination. But there are instances where no facility exists for taking advantage of this method, or where the silver and copper contents are not sufficiently high to warrant it, and then certain wet treatment may follow chlorination.

After the gold solution has been leached out of the ore, insoluble silver chloride remains behind. The pulp is therefore again lixiviated with a solution of hyposulphite of lime

or of soda, and the silver solution, when leached out, is precipitated by calcium polysulphide.

At the Plymouth Rock mill, California, according to T. K. Code,* this practice gives an extraction of 90 to 96 % of the gold, and 86 to 91 % of the silver. Also, after the gold has been thrown down, 60 % of the copper is recovered, either by electrolysis (when rich), or by precipitation with scrap iron (when poor). The total cost of treatment, on 5000 t., was *26s.* a ton.

The concentrates from the Idaho mill, Grass Valley, California, carry much lime and magnesia, some manganese, chalco-pyrite and zinc-blende, and 4 oz. gold and 8 to 12 oz. silver per ton, necessitating special treatment, which is thus described.† Roasting is done at black heat, without salt, on account of excessive volatilisation. When the charge is quite cold, it is wetted down and screened immediately to avoid agglomeration due to anhydrous sulphates. By 48 to 72 hours' leaching with cold water, the soluble sulphates of copper, iron, and zinc are removed, with a small proportion of the silver and gold (which are recovered in the copper tank). Alkaline bases and carbonates are next neutralised by adding 60 to 100 lb. per ton of sulphuric acid at 66° B.; then the oxide of copper is dissolved by introducing 20 lb. more acid (diluted) per ton, and agitating for 24 hours; and finally the charge is washed with cold water for 48 hours, shovelled out, partially dried, screened back into the tank, and impregnated with chlorine. After leaching, the tailings contain 4 to 6 dwt. gold and all the silver. To them, 5 % salt is added, and the mixture is dried and roasted as rapidly and fiercely as possible, "when little loss by volatilisation is experienced." Leaching with hyposulphite affords the silver. No details of cost have been published, but they must be excessive, there being two distinct roasts, a sulphating and a chloridising, and repeated handlings.

The concentrates produced at Brad, Hungary, contain iron pyrites, zinc-blende, barytes, antimonial minerals, argentiferous

* Mineral Industries for 1898, p. 344.

† En. & Min. Jl., Dec. 20, 1890.

galena, and some carbonate of lime and magnesia, with 15·35 dwt. gold and 4 oz. silver per ton. They are treated by Munktell as follows.—After an oxidising roast for 24 hours (to remove 36 to 40 % sulphur), 5 % salt is thoroughly mixed in with the ore, and 4 hours later the charge is drawn into a covered pit where it can cool slowly. Leaching is done with 5 different solutions. First, 250 gal. per ton of warm water dissolves out chlorides of copper and zinc, with about 25 % of the silver, the remainder of which is next recovered by adding 160 gal. per ton of 2 % solution of soda hyposulphite ; following this, iron oxides are removed by 270 gal. per ton of dilute sulphuric acid ; then weak solutions of bleaching powder and sulphuric acid dissolve the gold, and cold water washes out the solution. The whole process occupies 9 or 10 days. Sodium sulphide is used for precipitating the gold and silver, which are dried, pressed, and roasted ready for melting. The cost per ton is stated to be :—

| | *s.* | *d.* |
|---|---|---|
| 100 lb. salt .. .. .. .. .. .. .. | 1 | 5 |
| 91 lb. sulphuric acid .. .. .. .. .. | 5 | 4 |
| 22½ lb. bleaching powder .. .. .. .. | 3 | 3 |
| 13 lb. soda hyposulphite .. .. .. .. | 1 | 7 |
| Fuel .. .. .. .. .. .. .. .. | 7 | 1 |
| Labour .. .. .. .. .. .. .. .. | 3 | 3 |
| Total .. .. .. .. | 21 | 11 |

# CHAPTER X.

## CYANIDATION.

THOUGH the solubility of gold in cyanogen compounds was known almost a century ago, it is only within the last few years that practical application of the knowledge has been made in the recovery of gold from battery tailings and from ores direct by solutions of potassium cyanide (KCy), the operation being known as cyanidation.

**Chemistry.**—Volumes have been written on the results of investigations made, with more or less care, into the chemical reactions involved in the cyanide process ; but these reactions are so complicated that their study is by no means complete, and it is not necessary here to go beyond the main features having a bearing upon the practical application of the process.

*Equations.*—Neglecting the host of secondary reactions arising in practice from the composition of the ore and impurities in the leaching liquors, the commonly accepted formula representing the dissolution of gold by potassium cyanide is—

$$2\ Au + 4\ KCy + O + H_2O = 2\ AuKCy_2 + 2\ KOH\ ;$$

though some chemists prefer the notation—

$$4\ Au + 8\ KCN + O_2 + 2\ H_2O = 4\ AuCNKCN + 4\ KHO\ ;$$

and others again contend that the first reaction is—

$$\begin{aligned} &2\ Au + 4\ KCy + 2\ H_2O + 2\ O \\ &= 2\ AuCyKCy + 2\ KOH + H_2O_2. \end{aligned}$$

and that the hydrogen peroxide thus formed acts upon another part of the gold not yet dissolved in the first reaction, as follows—

$$2\ Au + 4\ KCy + H_2O_2 = 2\ AuCyKCy + 2\ KHO.$$

However this may be, the essential facts established are the dissolution of the gold by KCy in aqueous solution and in presence of oxygen, and the consequent formation of an auro-potassic cyanide salt and caustic potash.

*Action of KCy on Base Metals.*—But the solvent action of KCy liquors is not confined to gold; it is exerted on many other metals, both in their free state and when present as compounds—oxides, carbonates, chlorides, sulphides, sulphates, etc. The successful application of cyanidation to gold recovery from complex ores and tailings depends on an appreciation of the conditions under which a maximum dissolution of the gold with a minimum dissolution of base metals (involving waste of KCy) is favoured.

It has been shown by J. Mactear,* F.R.S.E., that the action of KCy solution on "freshly precipitated" oxides of iron, zinc, copper, and lead (those being the principal base metals in gold ores) is much greater than it is upon gold, and while such oxides are not encountered in nature, the treatment of ores by alkaline solutions in practical cyanidation has the effect of producing hydrated oxides which are readily soluble as indicated.

In the case of carbonates of these metals, the amount dissolved by a dilute as well as by a strong solution is very much greater than that of the gold or silver present with them, all the more so if oxygen be absent.

Any appreciable quantity of base-metal chlorides in gold ores is so rare that it may be neglected, but should chloride of silver be present, it is dissolved much more readily than metallic gold.

The base-metal compounds most commonly met with are the sulphides of iron, copper, lead, and zinc, in the form of

* Trans. Inst. M. & M., iv. 37 (1895).

pyrites, galena, and blende. In their case, considerable differences occur in the rate at which they dissolve, physical condition as well as chemical composition affecting it. Whatever the strength of the solution, its action is complicated by the presence of alkali as caustic or carbonate of potash, either present originally in the solution, or produced in it by the reactions taking place in the process. This alkali acts on the sulphides in certain cases, complicating the reactions, but as a general rule the amount of base metal dissolved is much greater than the gold or silver, even where oxygen is present.

Even water will dissolve most of the base-metal sulphates, without dissolving the gold. The presence of alkali in the solution also affects the result, as the soluble sulphates are decomposed, with the formation of carbonates or oxides, which are acted upon as described above.

Mactear considers it necessary to regard the question as involving the effect of electric action also. A mixture of particles of metals cannot be treated by KCy solution without bringing into play the action of the galvanic couples thus formed, nor indeed can sand containing particles of zinc and gold (although these be not in galvanic contact), be treated without formation of galvanic couples and consequent complication of the action, for as the gold enters into solution and is diffused it comes in contact with the zinc, which precipitates it, and thus forms the galvanic couple. If galvanic couples are placed in KCy solution, the negative metal will be protected from attack while the positive metal will be dissolved: precipitation of gold and silver from such solutions by aluminium, zinc, and copper is a good illustration of another form of this galvanic action.

Most, if not all, of the sulphides occurring in nature are negative to the metals. As to the practical effect of this galvanic-couple action, Mactear asserts that "there can be but little doubt that dissolution of the gold is considerably accelerated by it." Such galvanic couples may readily be formed in the case of gold ores by the gold coming in contact with small particles of metallic iron worn off the shoes and dies of the stamps, often amounting to 1 or $1\frac{1}{2}$ lb. per ton of

ore, a quantity enormously greater than the gold usually present in ore treated by the cyanide process. On the other hand, it is not found in practice that the presence of pyrites in tailings facilitates the treatment, nor is the extraction from concentrates better or quicker than from ordinary ores, but rather the opposite.

The action of the base-metal sulphides is in fact very obscure, and it is not safe to generalise in any particular concerning them. The various kinds of iron pyrites for instance behave quite differently, as has been abundantly shown by R. Recknagel,* and similar contradictions may be anticipated with the sulphides of copper, silver, and zinc. Then nearly all sulphurets are of mixed composition, introducing further complications. Hence, experimental treatment of each mine's product is the only reliable guide to what will happen.

*Selective action.*—There is, however, no doubt that if the action of cyanide solution is arrested immediately the gold has been dissolved, a dilute solution will be found to have dissolved—in the case of most ores containing sulphides—less of the base metal compounds than a stronger solution. This naturally follows, there being, even in the most dilute solution employed, an enormous excess of KCy over the amount required for the solution of the gold, the excess exerting its solvent action upon the base metal compounds: thus if a solution containing 2 parts cyanogen to 1000 of water be used to treat ore or tailings containing 15 dwt. gold per ton, there will be present about 160 times more cyanogen than is chemically required for the gold dissolved. Further, the amount of oxygen present in KCy solutions in the form of dissolved air or oxygen is greater in dilute than in strong solutions. From this cause, as well as from the greater mobility of weak solutions, it would be natural to anticipate that the action of the more oxygenated dilute solutions upon gold or silver would be greater than that of the stronger solutions; and the recent work of Maclaurin and others has placed this beyond doubt.

Herein lies the basis of the claim for a "selective action"

* En. & Min. Jl., Nov. 13, 1897.

on the part of very dilute solutions of KCy for gold, so that it passes into the liquor as auro-cyanide of potassium while the base metal-ores are left behind in the solid mass. The reverse is held to occur with strong solutions.

According to W. A. Dixon,* it is a question of oxygen, whether supplied by the air present in solution in the leaching water, or by special oxidising agents, such as bromine, chlorine, ferricyanides, peroxides, etc. He finds that a ton (2240 lb.) of ore in powder requires 100 to 110 gal. of water to thoroughly wet it, or an average of 105 gal., which, at 70,000 gr. per gal. is equal to 7,350,000 gr. The coefficient of solubility of oxygen at 30 in. barometric pressure and 60° F. is ·0295, which, divided by 5 to reduce it to that due to the partial pressure of oxygen in air, gives as the coefficient of absorption of oxygen from air ·0059, so that 105 gal. of water would contain 43,383 gr. measures of oxygen in solution, which would weigh 62 gr. These 62 gr. of oxygen contained in 105 gal., as a maximum, would, in conjunction with KCy dissolve 1527 gr. (3·181 oz.) of gold, and the quantity of cyanide required would be 1001 gr., so that the solution would be very dilute, containing only ·0047 % cyanogen.

Here, Dixon thinks, is a sufficient explanation of the "selective action." In presence of oxygen in all cases the affinity between soluble cyanides and gold is superior to that between soluble cyanides and compounds of base metals, while these compounds have the same affinity for cyanogen whether oxygen in the free state or easily available is present or not. The base metals always occur as compounds, whilst the gold is in the free state. In these circumstances, gold is first dissolved as long as free oxygen is available; then, oxygen being exhausted, base metals pass into solution until either they are exhausted or the cyanide is saturated, whichever happens first, a sufficient time being given to complete the reaction. A very dilute solution of cyanide contains much free oxygen in proportion to the cyanide; it therefore dissolves much gold in proportion to

* Trans. Inst. M. & M., vi. 88 (1897).

the cyanide present, and afterwards little base metal, because there is little cyanide left to saturate. "It therefore *appears* that the cyanide has selected the gold. In a solution containing more cyanide, gold is dissolved till the oxygen is exhausted, and then the excess of cyanide enters into double decomposition with the compounds of the base metals, which are then found in solution in greater proportion relatively to the gold, and the solution *appears* to have had a selective action on them." In practical work, the oxygen in solution in the water used might be much less than that indicated, being removed by organic and mineral substances undergoing oxidation; but, on the other hand, ores treated by the cyanide process have usually much less gold than 3 oz. per ton, or they are treated several times with fresh solutions, an oxygen-holding solution replacing one which is exhausted of that element.

Dr. Rose * combats this theory, finding it to be based on the assumption that "selective action," if it exists, must be absolute (that is to say, that one substance must be completely dissolved before some other can be attacked), which is quite erroneous. If two pieces of metal or other conducting substance of different composition are *in contact* in a liquid, then the one which has the higher solution-pressure (or is electro-positive to the other) in that liquid will be dissolved, and will protect the other from being dissolved; "but if they lie side by side *without touching*, both will be dissolved, each at the same rate as if the other were not present. As a matter of fact, the base metals (present as compounds) will be passing into solution all the time, the rate of dissolution increasing with the concentration of the solution, but having no connection with the presence of gold. This rate is certainly not increased by the absence of oxygen. Since gold cannot expel potassium from KCy, and some other substance, such as oxygen or bromine, must be present to assist in the removal of the potassium in order that gold may be dissolved, it follows that when such substances are present in large quantities the dissolution of gold is most rapid. Under these favourable

* Trans. Inst. M. & M., vi. 92 (1897).

conditions, gold is dissolved very quickly, even if the cyanide solution is so dilute that its action on the compounds of the base metals is slow. It is this state of things that is known as 'selective action.' In more concentrated solutions, the action on base-metal compounds being much more rapid, and that on gold being, under some conditions, slower than before, it results that selective action in favour of gold is less marked, and may even, from a practical point of view, be regarded as reversed." Dr. Rose also finds Dixon's estimate of the average amount of water required to saturate a ton of ore somewhat high. "The amount actually needed on dry ore varies from about 30 to somewhat over 50 %, or, say, 60 to 100 gal. per short ton (67 to 112 gal. per long ton). The average on unroasted ore would be well under 100 gal."

In the opinion of H. H. Greenway,* the equation

$$4\ Au + 8\ KCN + O_2 + 2\ H_2O = 4\ AuCNKCN + 4\ KHO$$

shows that for dissolution of gold, the following proportions by weight of the substances involved are necessary, viz. 197 parts gold, 130 KCy, and 8 oxygen. Obviously the desiderata are to dissolve the gold, to leave every other substance behind, and to use as little cyanide as possible. To leach a vat of crushed ore, he assumes that it is necessary to have about 700 lb. of solution per ton of ore to comfortably saturate and cover the ore to be treated. Water which has been exposed to the atmosphere, and is saturated with oxygen (as derived from the atmosphere), will contain about 8 parts by weight of oxygen in 1,000,000 parts by weight of water. The amount of KCy necessary to combine with this oxygen for the solution of gold is 130 parts, which gives a solution containing ·013 % KCy, 700 lb. of which solution will contain sufficient KCN and oxygen for the solution of about 2 oz. gold. "This then gives (under normal conditions) the *strongest* solution that can be of any avail for the solution of gold, unless more oxygen than that naturally contained in the water is made available for the reaction." Moreover, he

* Trans. Inst. M. & M., viii. 112 (1899).

contends that "in order to dissolve gold quickly (except when base metals are present), the solution should be as strong as possible, always bearing in mind that it is useless to have any KCN present unless a corresponding amount of oxygen is *present in solution* in the water; but as regards the base metals (with the exception perhaps of cadmium), no such limitation obtains, and the stronger the solution of KCN, the more rapid will be its action, so that a solution of any strength will have free play to dissolve these base metals under the conditions that obtain in an ordinary leaching operation." Further, there "is no such thing as an absolute solution of the gold and an absolute leaving behind of the base metals when a dilute solution of KCN is applied to a refractory ore. No matter how dilute the solution is made, *some* of the base metals will be dissolved together with the gold. In an ordinary solution containing (in solution) 8 parts oxygen per million and ·013 % KCN, the action upon the gold and the base metals is relatively proportionate, and follows the general rule that the stronger the solution the more rapid its action; but where there is in the solution a proportion of KCN in excess of that required by the oxygen in the water, the rapidity of its action upon the base metals increases, whilst its action upon gold is not quickened, because there is not sufficient oxygen available for the reaction." Accepting these inferences as correct, the fact is arrived at, that a "model solution" of KCy is one that (presuming the solution to be able to contain 8 parts oxygen per million) throughout the operation always contained ·013 %. On account of decomposition of KCN, and because some extraneous matters are always dissolved in addition to the gold, it is impossible to do this with any great accuracy, so that allowance has to be made to meet the exigencies of each particular case; but it is certain that if cyanide operators bear these facts in mind, the treatment of the various ores that come under their notice will be to a great extent facilitated."

The same authority claims to have discovered that a "saturated solution" of KCy has a far greater selective action than is possessed by any other strength of solution, the maxi-

mum extraction of gold and minimum "consumption" of cyanide being thereby achieved.

In an experimental treatment of ore from the Monowai mine, New Zealand, which carries 3·78 % copper pyrites, 4·4 iron pyrites, ·25 galena, ·26 zinc-blende, and ·13 alumina, and which cannot be treated economically by the ordinary cyanide process, when using a saturated solution "the consumption of KCy was so small that it could not be determined. From these results it would appear as though there were after all a solution of KCN possessing a perfect selective action, so as to dissolve gold and silver from ores, and leave the baser metals *absolutely* untouched, and that, strange to say, this is not a very dilute solution, but a saturated one." Other operators have thrown doubt upon the reliability of these conclusions, and their utility from a practical standpoint. Thus Maclaurin has shown that while a ·01 % KCN solution dissolves oxygen at about the same rate as ordinary water, this power increases up to ·25 or ·3 %, and after that it diminishes.

*Sources of Oxygen.*—As to sources of oxygen, Dr. Loevy* has expressed a positive opinion that the only economically available source is the atmospheric air, and that the commercial success of the process depends on the manner in which it is supplied. This dogma, while probably acceptable so far as regards the conditions of cyanidation on Transvaal tailings, and for many other cases of a simple character, does not apply in all instances. Prof. Christy,† while believing that, with low-grade ores and dilute liquors, the KCy solution itself will, if properly aërated, carry sufficient oxygen for dissolution of the gold, admits that, when some reducing agent is present in the water or the ore causing absorption of oxygen, and with rich ores and strong solutions, "there is sometimes a distinct advantage in the use of oxidising agents." The necessity that working solutions shall contain their maximum of oxygen is urged by W. Bettel,‡ and an

* Pres. Address Chem. & Met. Soc. S. Africa, Aug. 22, 1898.

† Trans. Amer. Inst. Min. Engs., xxvi. 739 (1896).

‡ Jl. Chem. & Met. Soc. S. Africa, i. 276 (1898).

excess of oxygen is found by A. F. Crosse* to have an "extraordinarily favourable influence"; while D. A. Louis† considers it an inconsistence "to expect any simple aqueous solution of cyanide to contain sufficient free oxygen for the purpose, inasmuch as water, at a mill, at ordinary temperatures, will not contain more than 96 gr. of free oxygen per ton, and this, under the conditions of extraction, will be utterly inadequate for the realisation of the reaction on a practical scale."

Hence, on the Rand, it was at one time the practice to force air to the bottom of the leaching-tanks, and allow it to bubble up through the charge, but it was found that the same practical aëration at much less cost was secured by simply circulating the pulp, pumping it from the bottom of one tank and letting it enter the top of another, some air being at the same time drawn into the suction pipe. (See p. 536.)

*Temperature.*—The temperature of the leaching solution plays an important part in the process, to which but little attention is given as a rule. But Dr. Loevy found in an experimental treatment of a pyritic ore assaying about 4 oz. gold per ton that he got a 14 % higher extraction by 4 hours' leaching at 104° F. than by 50 hours' leaching in the cold, and he states it as a fact that a temperature of 95° to 104° F. accelerates and increases the dissolution of gold to the extent that an additional extraction of 1 dwt. per ton may be obtained from tailings assaying 6 dwt. W. J. Sharwood‡ confirms the good results of raised temperature. Similarly, H. G. Heffron, working at the Sacramento mill, Mercur, Utah, proved the gold of that ore to be most readily dissolved at 95° F.; below 80° F., there was considerable falling off in the energy of the solution, and above 95° F. there was no gain.§ This advantage from increased temperature is most notable in ores with talcose gangue, and in those containing silver in large proportion. Thus, Butters intends using hot solutions in some Mexican plants he is erecting.

* Jl. Chem. & Met. Soc. S. Africa, i. 88 (1898).

† Trans. Inst. M. & M., iii. 240 (1895).

‡ En. & Min. Jl., Nov. 13, 1897. § Mineral Industry for 1898, pp. 327–8.

On the other hand, it is pointed out by A. F. Crosse that the heating should be applied to the ore rather than to the liquors, as the latter absorb more oxygen in the cold. A. von Dessauer got much higher extraction at the May Consolidated mill from sun-baked tailings than in cold weather. But J. R. Williams and F. F. Alexander attribute better extractions in summer to purer water (more free from organic matter).

*Loss of KCy.*—Causes of loss of cyanide in working may be classed under five headings, viz.

Consumption in dissolution and precipitation of the gold ;

Destruction by "cyanicides" in the ore ;

Destruction by organic matters occurring as mechanical impurities in old tailings ;

Absorption by leaching vessels ;

Waste in weak liquors discarded.

The first is as a rule a very small matter, and dependent chiefly upon duration of contact between gold solution and precipitant.

The second is by far the most important of all. Many ores require a preliminary treatment to remove soluble and insoluble substances which decompose KCy solutions, and some are of a nature to prohibit cyanide treatment.

Loss by organic matters in old tailings may be due to accidental and preventible occurrence of leaves, roots, etc., capable of elimination by screening or straining ; or to the deliberate presence of charcoal derived from heap-roasting of ore (see p. 278) or from barrel amalgamation of blanketings (see p. 255), which cannot practicably be removed.

Absorption occurs in new wooden leaching vats, but depends much upon the kind of wood used, and becomes trifling in time.

Loss in superfluous weak liquors run to waste is a variable quantity, controlled to a great extent by the nature of the ore, which governs the point at which impurities in solution prevent further regeneration of the exhausted liquors.

At one time, great loss of KCy was ascribed to oxidation by the air (converting cyanide into cyanate and then into

carbonate), absorption of carbonic acid from the air (liberating hydrocyanic gas), and deliquescence (resulting in a similar effect); but it has been conclusively shown by Dr. Wells * and Alfred James † that such losses are in practice quite insignificant.

**Applicability.**—The questions to be determined in judging whether cyanide treatment is applicable to any particular ore rest mainly in the ore itself—its physical and chemical characteristics—and in the condition of the gold carried by it. These considerations all have a commercial basis—the necessity and relative expense of wet or dry milling, whether coarse or fine crushing will serve, and whether calcination may be necessary or beneficial and at what cost. Time of treatment and capacity of plant must be duly taken into account. The nature and volume of available water for leaching is also a factor.

Investigations therefore have to be directed firstly to the ore and secondly to the liquors, to determine both the extraction and precipitation of the gold and the consumption or loss of KCy.

*Testing Ores.*—Some idea of the character of a raw ore may often be formed from its appearance: whether it is friable, clayey, talcose, oxidised, or pyritic; whether there are any incrustations of metallic salts present, or any indications of copper, iron, zinc, lead, or other metals, and the forms in which these occur.

Tailings and concentrates may be panned to ascertain the presence of coarse or fine gold, pyrites, or amalgam, as well as fine charcoal from drying or roasting processes.

Ores are usually milled to pass a 30-mesh sieve; extraction tests are made at this fineness, and then compared with results obtained by still finer reduction, bearing in mind that a laboratory grinding to 30 mesh leaves the product coarser and more granular than stamp-milling with screens of the same size.

* En. & Min. Jl., Dec. 14, 1895.

† Trans. Inst. M. & M., iii. 409 (1895).

Not infrequently it is found that contact with cyanide solution for a longer time gives an extraction equal to that obtained by finer grinding and shorter contact; in this case, the problem is solved by commercial considerations of size of plant and amount of ore to be treated. It is a very notable fact that in the case of friable oxidised ores, of not high grade, good extractions have been obtained without any grinding whatever, the stuff being put through a fine breaker, and sifted. Thus at Mercur $\frac{3}{4}$-in. gauge is often used, though better results are got at $\frac{1}{4}$ to $\frac{1}{2}$ in. On the other hand, extreme comminution is sometimes demanded, as at the Republic mine, Washington, where the ore is crushed to pass 120-mesh screens.

Tailings and concentrates are almost invariably tested in the same condition as received, it being recognised that re-grinding is more expensive than longer treatment, or even stronger solutions.

The general practice, which has been well described by Alfred James,* of the Cyanide Plant Supply Co., is to agitate 1000 gr. of the sample with 500 gr. of ·5 % solution in a 12-oz. corked bottle, which is placed in a box on shafting and left to revolve slowly overnight. On the following morning, the solution is filtered off; the consumption of cyanide is ascertained in a measured portion of the filtrate by titration with $\frac{N}{10}$ $AgNO_3$ standard solution; and, after the sample has been washed, the total remaining filtrate and washings are evaporated to dryness with litharge, to determine the amount of gold in solution. The sample thus treated, now termed "residues," is assayed, and the amount of gold recovered from the solution compared with that found in the residues gives the extraction, which is checked by an assay of the same sample before treatment. In evaporating the solution to dryness with litharge, it is advisable to add the litharge prior to commencing the evaporation, widely divergent results having been obtained in consequence of the neglect of this precaution.

* Trans. Inst. M. & M., iii. 369 (1895).

A. F. Crosse and others have proposed to avoid evaporation, by treating solutions direct with an excess of silver nitrate, melting the resulting insoluble precipitate of silver and gold cyanide, and parting the cupelled button.

If the cyanide consumption has been heavy, say, over ·15 %, it is usual to test 1000 gr. of the sample for acidity, with a standard soda solution made up of such a strength that each c.c. of soda solution per 1000 gr. of ore tested equals ·1 lb. of commercial caustic soda to be added per ton of ore. "Percolation" tests are then made, using an alkaline wash, or water and alkaline washes, prior to lixiviation with cyanide. If soluble salts are present, they should be removed by water-washing prior to the alkaline treatment. On the working scale, with ores containing acid salts, lime is added as a neutralising agent to the stamp-box, and gets thoroughly mixed with the ore while being crushed. To leach first with an alkali finely-crushed ore interferes with filtration, and, through the settling of the ore, frequently prevents an even percolation and equal exposure to the cyanide solution. In some cases, where caustic soda or lime has not been efficacious, a preliminary acid wash has been given, followed by a water wash and then an alkaline wash; in exceptional cases, this method has been attended with some success. Usually, however, the results are not improved; on the contrary, the acid treatment seems to break up the refractory compounds, and make them more destructive to cyanide.

Lime is usually accounted preferable to caustic soda for neutralising acidity, because it does not lead to formation of zinc ferrocyanide, which impairs the activity of the zinc used for precipitation.

The occurrence of a blue coloration in crevices of the vessels, in the liquors, or in the ores, is in itself evidence of acidity and of the consequent destruction of KCy, Prussian blue resulting therefrom. Sometimes the scum on the liquors shows whitish until exposure to air and light has completed the reactions.

Where the consumption of cyanide has been low, but the extractions have not been good, longer period of contact,

increased fineness of grinding, altered proportion of cyanide and solution to ore, and agitation instead of percolation, may improve the results, and tests are therefore made on these lines. In some exceptional cases it is found that the extraction increases exactly in the ratio of consumption of cyanide; thus a consumption of ·01 % KCy gives 20 % extraction, and ·04 % KCy gives 80 % extraction, increased consumption being caused by longer contact. Again, in some obstinate cases of cubical iron pyrites in a silicious gangue, agitation of the ore-mass by jets of compressed air gives a fairly good extraction.

In the presence of considerable amounts of hydrous silicates, as well as when sulphides of silver, copper, lead, iron or zinc are present, good extractions have been obtained from roasted ores which could not be economically treated in the raw state. Some cupriferous and other ores have been dealt with by taking advantage of the selective action of dilute solutions; but there are still classes of ore excluded, and on some of these roasting at a low heat is attended with considerable increase of yield. Thus certain of the oxidised ores of Mercur, Utah, proved most difficult to treat on account of their talcose and argillaceous character, due to hydrous silicates of magnesia and alumina; but after dehydration, by roasting just sufficiently to expel the combined water, they leach freely.

At the Republic mine, Washington, three representative samples assayed as follows:*—

| | *a.* | *b.* | *c.* |
|---|---|---|---|
| Silica | 85·61 | 90·55 | 94·41 |
| Alumina | 6·37 | 4·73 | 2·95 |
| Iron oxide | 3·16 | 2·57 | 1·29 |
| Copper | — | — | ·016 |
| Zinc | trace | — | ·025 |
| Manganese | — | ·20 | trace |
| Lime | — | ·90 | ·06 |
| Lime carbonate | 4·37 | — | — |
| Sulphur | ·04 | — | ·06 |
| Water | — | 1·20 | 1·14 |

The hydrated oxides of alumina and iron seem to envelop the gold and prevent its dissolution by the cyanide. The

* En. & Min. Jl., April 28, 1900.

lime carbonate sometimes present is proved to be not responsible for any difficulty in treatment. A dehydrating roast leaves the oxides in a porous condition, and renders cyaniding quite easy; but the silver extraction is reduced by the operation.

Much the same result has been experienced with the Kalgurli (W. Australia) ores, which always contain carbonates of lime and magnesia (sometimes $1\frac{1}{2}$ % of the latter), and some easily-decomposed silicates, as well as soluble aluminous salts, and a dehydration roast has been found most beneficial. But this calcination must be at a temperature not approaching redness, in fact it must not exceed 300° F. If such ore is heated on the large scale to temperatures approaching redness, a calcined mass is obtained, which, when moistened with water, becomes similar to hydraulic cement, setting into hard solid masses. Very careful comparative percolation tests were made by H. L. Sulman on masses of this ore (*a*) before heating, (*b*) on heating to 256° F., and (*c*) after being heated to redness. In the case of (*c*) the ore was found to percolate slowly for a time, but then to "set" to an almost solid mass; while in (*b*) the percolation was very much better than (*a*), being continuous and uniform.

Some of the Mercur (Utah) pyritic ores contain native sulphur, realgar, orpiment, and ferrous arsenide, all of which affect the solution. The alkaline KCy acting on the sulphur and sulphides of arsenic produces a soluble alkaline sulphide, which absorbs all the oxygen and thereby prevents dissolution of the gold. Sometimes also a carbonaceous shale occurs, and its carbon causes precipitation of gold from solution. The supposition that the arsenic is the objectionable element of the ore is controverted by W. Orr,* who states that ore carrying $3\frac{1}{2}$ % arsenic in the minerals scorodite and haidingerite has been treated successfully; in fact arsenic is always present in the KCy solutions at Mercur, and is precipitated to some extent with the gold on the zinc. He attributed all the trouble to the sulphides and carbon, and this is disposed of by a preliminary oxidation roast.

* Proc. Internat. Min. Congress, Salt Lake City, 113-7 (1898).

It is of the utmost importance to discriminate between the two roasts—dehydration and oxidation—and in the case of the latter to ensure it being complete, as a partial roast resulting in sulphates is most undesirable. A sulphating roast, with a view of leaching out soluble sulphates, is never adopted, because the ideal effect is not obtainable in practice.

Experimental treatment by A. C. Claudet * of an ore consisting of hydrated oxide of iron and silicious rock, from Hannan's, W. Australia, gives interesting evidence of the effect of calcination. Analysis of the ore showed :—

| | % | | % |
|---|---|---|---|
| Iron peroxide . . . | 15·45 | Lime . . . . . . | ·20 |
| Manganese oxide . . | ·18 | Alumina . . . . | 1·66 |
| Copper . . . . . . | ·05 | Silicious rock . . . | 77·93 |
| Sulphur. . . . . . | ·12 | Water and loss . . | 4·41 |

With about 3 oz. gold per ton.

The silicious rock contained :—

| | % | | % |
|---|---|---|---|
| Silica . . . . . . | 56·60 | Magnesia . . . . | ·25 |
| Alumina . . . . | 16·92 | Alkalies, chiefly potash | 3·66 |
| Lime . . . . . . | ·30 | Loss . . . . . . | ·20 |

On another sample of the same ore assaying about 2½ oz. gold, using ·25 % KCy solution, it was found that, operating on raw ore, the consumption of cyanide was 30 % and extraction was 88·4 %; after a dehydrating roast, the KCy loss was only 10 % and the extraction was 94·6 %. In each case, there was 48 hours' contact.

According to Alfred James,† typical Kalgurli ores have the following composition :—

| | % | | % |
|---|---|---|---|
| Silica, about . | 50 | Zinc, about . . . . | 0·2 |
| Iron ,, . . | 10 | Arsenic . . . . . . | traces |
| Alumina . . . | 5 to 20, or more | Antimony, about . . | ·02 |
| Magnesia . . | 1 to 5 | Tellurium . . . . . . | ·03 to ·1 |
| Sulphur . . . | 3 to 7 | Calcite . . . . . . | 6 to 17 |
| Copper . . . | ·1 to ·3 | Sulphates of sodium, magnesium and aluminium | about 2 |
| Lead . . . . | traces | | |

the sulphur occurring as sulphide being about 4 %, and the iron pyrites 3 to 7 %. By roasting these ores, a notable im-

* Trans. Inst. M. & M., v. 327 (1897).

† *Ibid.*, viii. 486 (1900).

provement in extraction was secured by James, but he omits to give precise details of how the roast was conducted, which is a vital point.

The subject will be again referred to when discussing practical operations on sulphuretted ores.

After a general idea of the behaviour of the sample has been gained, it is usual to make all tests by percolation, in view of the adoption of that type of plant when possible. In certain exceptional cases, however—usually with concentrates or certain silver ores—agitation is found to be practically indispensable, and in this event the larger tests are made in revolving barrels, or vertical stationary cylinders provided with agitators, which latter are much to be preferred.

Percolation tests are made in bottles, lamp-glasses, half-barrels, or larger tubs, according to the amount to be tested; a false bottom is constructed in the apparatus employed, the dry ore is laid on this, and the washes and solutions are added.

Before proceeding with the erection of a plant, it is usual to confirm the small tests referred to above by experiments on larger quantities, with charges of ½ to 3 t.; but laboratory tests, if carefully and intelligently made, form a most useful and reliable guide to actual practice, so much so that it is not unusual for large contracts to be made and purchases of tailings to be effected on the treatment of samples of less than 5 t. One factor, however, requires to be borne in mind; it is that the consumption of cyanide is invariably greater in test experiments than in practice. This is usually accounted for by the plant becoming "saturated" in regular working, whilst the test apparatus is used intermittently, and to the fact of there being a greater loss in small operations; but Alfred James is inclined to consider that no small proportion of the lessened consumption is due to the regeneration of cyanide in the exceedingly complex solutions resulting in daily practice from the continuous circulation of the liquor in the works. On a New Zealand ore, he made it a practice to calculate on the consumption in actual work of only ⅓ of the amount indicated by laboratory tests, and this estimate was shown in

practice to afford a very safe margin. In Africa, when applying cyanide lixiviation to the treatment of tailings, laboratory tests showed an average consumption of $3\frac{1}{3}$ to $4\frac{1}{2}$ lb. of cyanide per ton; this divided by 3 gives $1\frac{1}{3}$ lb. per ton, which is also a safe margin above the actual consumption in practice at the mines from which these tailings were obtained.

On the other hand, it is not at all unusual for extractions to be obtained regularly in actual practice of a higher percentage than those indicated in the laboratory. As a general rule, therefore, one is quite safe in estimating on the extractions yielded in experimental tests; but too great assurance is not warranted where copper or antimony is present.

It is very hard to define the limit to which ores may be profitably treated by cyanide; actual tests on each ore are necessary, but those with heavy coarse gold must be separately treated for its recovery either prior or subsequent to lixiviation. Ores with a friable gangue not unusually yield better extractions than those in which gold or pyrites is in very hard quartz; whilst very decomposed concentrates, and ores with incrustations of soluble metallic salts, do not lend themselves to commercially successful treatment. In the case of concentrates and tailings containing amalgam, means are provided for collecting the amalgam, which readily settles out after lixiviation.

Most commonly, tailings may be better treated as they are delivered from the battery, thus saving extra handling and avoiding acidity by the oxidation of contained pyrites; but sometimes better results are obtained after allowing the tailings to weather, this being notably the case at the Coromandel mine, India.

Similar experience is recorded by McCormick at Ouro Preto, Brazil, with vanner concentrates. The best results were got when allowing them to drain for 3 days after removal from the boxes, by which time they were "dry and porous, and the cyanide solution quickly attacked the gold." With perfectly fresh concentrates from which the water had not escaped, most unsatisfactory results were obtained.

A large percentage of sulphide of silver, which dissolves

so slowly as to practically baffle cyanidation, is the great difficulty with some New Zealand ores. But it may be stated in general terms that silver occurring in oxidised surface ores, or as chloride, is readily attacked by cyanide, and that, where other conditions permit, cyanidation may be economically applied, though, even with these conditions, it has, in Janin's opinion, a limited range of usefulness. Results obtained with different samples from the same mine exhibit wide variation. As a rule, the time required for treatment and the consumption of KCy prevent commercial success, though Packard says several Arizona plants are working on silver ore. The bullion produced by the New Zealand mines is about $\frac{1}{5}$ to $\frac{1}{6}$ gold only ; and the Comstock tailings now being successfully treated contain more than half their value as silver.

Copper compounds, when in a hard and resisting condition, are not much affected by KCy, and some ores carrying copper pyrites are successfully cyanided; but when the copper salts derived from the ores or resulting from pan amalgamation are soft, cyanidation is precluded, however dilute the liquor used, copper passing into solution and accumulating till it is precipitated on the zinc. In some cases, perhaps, the evil may be overcome by leaching out the copper as sulphate, and recovering it as a bye-product, taking care to very thoroughly wash out remaining traces of sulphuric acid before cyaniding.

Antimony in any quantity is a serious bar, causing excessive consumption of KCy or very fine grinding in order to dissociate the gold.

Much the same remarks apply to bismuth, which, however, is a metal of less common occurrence. The sulphide in each case is the most objectionable form of the metal.

Cobalt and nickel, both comparatively rare in auriferous ores, are said to behave in the same way as copper.

Manganese ore (pyrolusite) has been suspected of oxidising cyanide to cyanate, but not proved to do so.

Presence of tellurium would seem, according to Dr. T. K. Rose, to be positively beneficial. On the question of tellurium being dissolved by cyanide, and precipitated by zinc, he

points out that "if tellurium is alloyed with gold and treated with cyanide, it has a quickening effect on the dissolving action of the cyanide; the gold appears to be positive to the tellurium, and is dissolved more rapidly in consequence of the presence of the tellurium, and none of this element is attacked as long as any of the gold remains undissolved in contact with it." But there may be a difference in effects between laboratory tests on an artificial alloy and working tests on complex ores. Tellurium oxide is asserted to be insoluble in KCy.

*Testing Solutions.*—The consumption of cyanide is ascertained by titration with the standard $\frac{N}{10}$ solution of $AgNO_3$. If this solution is used in a works, the foreman measures out 13 c.c. of solution to be tested instead of 10; and the number of c.c. of standard solution consumed divided by 10 gives the percentage strength in KCy of the solution. More usually, however, a special solution is made, containing 13·08 grm. of pure triple-crystallised $AgNO_3$ per litre; 10 c.c. are then tested instead of 13, as with the $\frac{N}{10}$ solution, and the strength is read off from the burette as before.

Cyanide is here referred to in terms of 100 % KCy. In actual practice, cyanide is used containing 70 to 98 % of cyanide estimated as the potassium salt; but the higher grades usually contain a considerable amount of cyanide of sodium, which, when pure, contains 53 % of cyanogen, as against 40 % only in cyanide of potassium. Hence a sample of the pure sodium salt reported in potassium terms would appear to contain 132·5 % KCy.

A matter of the highest importance is the effect on gold extraction produced by continued re-use of cyanide solutions. Such solutions contain, amongst other matters, salts of zinc, iron, and copper, alkalies and alkaline carbonates, ammonia, and sulpho-cyanides; and unless these constituents are prevented from accumulating in the solutions, or their action can be greatly diminished by some means, the solvent power on the precious metals of any added cyanide must sooner or later be impaired. The alternative is, of course, to use such

weak solutions of KCy and obtain such complete precipitation of bullion as to admit of running the used liquors to waste. This, however, has seldom been found possible in general practice, and in the great majority of cyanide plants now working, the same liquors continue in use for years, subject to the usual additions and subtractions, fortifications and exhaustions.

Numerous experiments made by Alfred James * go to prove that extractions are always less (both on ore and on tailings) with fortified old solutions than with the same strength solutions made with fresh water ; and that addition of lime remedies this only in extractions from material containing practically quartz and gold alone, while it is actually harmful when sulphides are present.† Constant testing of these solutions is therefore imperative. James finds that, though a universal law cannot be laid down for the best treatment of such sump-solutions, the addition of lime, with time for the subsidence of any precipitate formed, is of advantage in the case of very free and coarse gold ores ; and that treatment with sodium sulphide, care being taken to avoid an excess, followed by the addition of a small amount of lead salt in excess (such as acetate or chloride) is efficacious where lime fails, time being given for any precipitated sulphides to separate out and subside. By such treatment, the necessity for running foul sump-solutions to waste may be avoided, when water is scarce, or the cyanide or gold contents of the liquor are high, and the gain by treatment is about $7\frac{1}{2}$ %. It is in practice on the Rand (at the Crown Reef) with most beneficial results.

The usual practice of precipitating gold from auro-cyanide solutions by zinc, causes accumulation in the solutions of a quantity of zinc salts. The suggestion that the double cyanide of zinc and potassium in the presence of caustic

* Trans. Inst. M. & M., vi. 2 (1897).

† On the other hand, Professor A. K. Huntington reports that "when the solution has already taken up a certain amount of gold, it seems to be in a condition to dissolve gold much more rapidly," which "shows that there is a real utility in using the solutions over and over again, without extracting the whole of the gold from them." Trans. Inst. M. & M., iii. 253 (1895).

potash splits up into a simple cyanide plus oxide of zinc and potassium—

$$K_2ZnCy_4 + 4KHO = 4KCy + K_2ZnO_2 + 2H_2O$$

is shown by James, by means of experiments involving the crystallising out of the products, not to hold good, and that, on the contrary, any oxide of zinc and potassium in solution combines with the added cyanide of potassium to form double cyanide of zinc and potassium, thus—

$$K_2ZnO_2 + 4KCy + 2H_2O = K_2ZnCy_4 + 4KHO,$$

which indicates that the zinc oxide in solution actually takes up the cyanide added by the operator to make the solutions of normal strength, though the silver test leaves him in ignorance of what has happened. This is one of the drawbacks of the zinc precipitation process as compared with others which leave the solutions in a condition of greater energy. The accumulation of zinc in solution is, however, moderated by the action of the sulphides contained in the ore and in the cyanide, and treatment with sodium sulphide and lead salt, as described above, completes the remedy.

On the authority of Packard, the arsenic of the Mercur ores soon contaminates the sump-solutions to such an extent that they cannot be re-standardised, though Orr says (p. 488) the arsenic does not interfere.

*Testing Water.*—It is by no means safe to assume that the main water supply which is going to be used on a working scale will be as free from objections as the distilled water or rain water perhaps employed in laboratory tests, and this is a matter demanding close attention. Organic matters in solution or suspension are frequently present in surface waters, and mineral salts are often dissolved in underground water, as also in the lake water of some districts. At the same time, water charged with salts will not necessarily and invariably have a bad effect on cyanide extraction. For instance, alkaline salts seem to be practically innocuous with ores carrying simple quartz and gold, and the presence of alkalies in a caustic condition will even increase the extraction in such a

case. But they are distinctly undesirable when sulphides exist in the ore. So also are salts of magnesia and alumina, which must be counteracted by addition of lime. Thus the water at Kalgurli, W. Australia, has a rapid action on KCy, a ·25 % solution falling to ·18 % in 20 to 30 minutes, and in a ·1 % solution the diminution of cyanide contents is very marked. On adding lime, this loss immediately ceases almost entirely. From experience at Bodie, California, it would seem that in some cases an excess of alkali over that theoretically required to counteract acidity has a beneficial action.

A plant with a capacity of 1500 t. a month, at Taltal, Chili, is compelled to use sea-water for making solutions and for washing, and it answers as well as fresh water. On starting, a rather heavy precipitate of magnesium hydrate and carbonate was formed by the caustic soda; but as the quantity of sea-water added is comparatively small, this precipitate becomes hardly noticeable. In consumption of cyanide there is practically no difference between sea-water and fresh water. The plant operates on tailings of 6·37 dwt. value, extracts 80 %, and consumes 1 lb. KCy per ton.

Organic matter in water may, however, interfere most seriously with precipitation, and cause heavy loss of gold.

*Estimating Oxygen in Liquors.* — A modification of Thresh's method (see Sutton's 'Volumetric Analysis') used most successfully in the Transvaal by A. F. Crosse,* is as follows. All cyanides and absorbents of iodine are first removed by precipitation with zinc sulphate, the amount required being determined by titration of a portion of the solution with a standard zinc-sulphate solution, using phenol-phthalein as an indicator, and running the zinc-sulphate solution into the KCy solution until the magenta colour of the latter is just destroyed. The proper amount of zinc-sulphate solution having been added to the KCy solution under examination, and the zinc cyanide having been allowed to settle, the clear liquor is siphoned off without due access of air, the end of the immersed leg of the siphon being covered with a small bag of lint to filter off any floating particles of

* Jl. Chem. & Met. Soc. S. Africa, i. 59 (1898).

precipitate. About 300 c.c. (2 or 3 pipettes full) of the siphoned solution are retained. A preliminary test of the iodine-absorbing power of the solution due to unprecipitated double cyanide is made by adding to a quantity equal to that used in the test ·9 c.c. of dilute sulphuric acid (1 : 1) and a few drops of potassium iodide and starch. Dilute bromine water (1 Br : $2H_2O$) is added until a blue colour is obtained. Another pipetteful of the liquid is then taken, ·9 c.c. of sulphuric acid and the required amount of bromine water found by the preliminary experiment are added, the stopper is put into the wide-mouthed bottle used in Thresh's test, and the pipette is turned over several times ; 1 c.c. of the potassium iodide and sodium nitrate solution is then added, and the free iodine, freed in proportion to the oxygen in the solution, is determined by means of standard sodium thiosulphate. The following reactions occur in the pipette, and after the addition of the thiosulphate and starch solutions :

$$(a)\ H_2SO_4 + KNO_2 = KHSO_4 + HNO_2.$$
$$(b)\ 2HNO_2 + 2HI = 2H_2O + I_2 + N_2O_2.$$
$$(c)\ N_2O_2 + H_2O + O = 2HNO_2.$$
$$(d)\ HCy + I_2 = ICy + HI.$$
$$(e)\ 2Na_2S_2O_3 + I_2 = 2NaI + Na_2S_4O_6.$$
$$(f)\ HI + ICy + 2Na_2S_2O_3 = HCy + 2NaI + Na_2S_4O_6.$$

A correction has to be made for the small quantities of nitrites in the cyanide solution. To make this correction, pour into it, in a very strong 350-c.c. flask, a quantity of solution equal to that used in the experiment (say 293 c.c.), add a few drops of potassium hydroxide, and close the flask with a rubber stopper having one perforation, through which is passed a glass tube with a glass stopcock. Boil the solution for a few minutes, and close the stopcock. Cool the flask ; when cold, pour the liquid into the pipette, and add 1 c.c. of iodide and nitrite solution and 1 c.c. of sulphuric acid (1 : 1). Then let it stand for 10 minutes, and, in the presence of coal gas, run it into the wide-mouth bottle employed as above, add starch, and titrate with thiosulphate. The quantity required gives the correction for nitrites and for the reagents,

as the same amount of acid and of iodide and nitrite solution is used in each case.

*Wet* v. *Dry Milling.*—Stamp battery tailings are of course always the product of wet milling, and, as their treatment by cyanide is in the nature of a supplementary process, no question arises as to the relative advantages of dry and wet milling. With ores treated in bulk, however, the case is quite different, and a vast amount of discussion has taken place on the problem whether the wet or the dry method of reduction gives better results. The term "better results" embraces a good many factors outside of actual cyanidation—such as potential output of wet and dry milling respectively, facilities for and cost of drying, in addition to those connected directly with the process—to wit, volume of solution, amount of slimes produced, and so on.

Wet crushing for direct cyaniding, using weak cyanide solution instead of water in the battery, is at once met by the very grave objection that at least 6 times (sometimes rising to 10 times) as much water is employed as is necessary for the leaching operations ; there would thus be 6 to 10 times as much cyanide liable to decomposition though not performing any useful duty, and a consequent greatly increased consumption of that article. In addition, there would be a corresponding multiplication of the amount of gold circulating in the solution ; and the solution itself would be diluted in the same proportion, increasing its bulk to a most inconvenient degree, and adding to the difficulties of precipitation.

Then again, for many ores, fine crushing is an absolute necessity. In carrying out some experiments in New Zealand to ultimate extraction point (95 %), Alfred James* found it compulsory to pass every particle through a 90-mesh screen, and he considers that "would be practically impossible with wet crushing, because with screens so fine the output would be very much less in comparison with that by dry crushing." No direct evidence is tendered on this point by him, and it is more than doubtful whether this statement can be substantiated. James only cites as a confirmation of it

* Trans. Inst. M. & M., vii. 41 (1899).

that when the Crown mine crushed dry their extraction was 90 to 93 % and their output about 1·34 t. per head per diem; but when they tried crushing wet, their extraction fell to 85 % and their output did not increase.

Manager Freeman, of the George and May mill, adopted dry crushing (through a No. 3 Gates breaker) and direct cyaniding in 1895. The washes used were:—1st, 25 t. of ·06 % KCy; 2nd, 25 t. of ·28 %; 3rd, 40 t. of ·1 %; 4th, 40 t. of ·05 %; finally, 5 to 10 t. clean water. Treatment occupied about 60 hours. Consumption of KCy, ·89 lb. per ton treated, some lime being added. Ore assay before treatment, about 5¼ dwt.; actual recovery, 4 dwt.; tailings assay, 1 dwt. The results for one month showed an extraction on ore assaying 5¼ dwt. of 76·2 %, leaving a profit of about 3*s.* 9*d.* per ton. The same ore treated by amalgamation and cyaniding would have given a yield of 12*s.* 3*d.* per ton (as against 15*s.* 9*d.* by direct cyaniding), at a cost of 16*s.* per ton (as against 12*s.*), or a loss of 3*s.* 9*d.* per ton as against a profit of 3*s.* 9*d.* by direct cyaniding.

At the Afrikander mill, when dry crushing and direct cyaniding a hard pyritic rock, the extraction was 86 % while the crushing machinery was new, but fell to 72 % when it had worn to such an extent that 27 % of the pulp would not pass ¼ in. mesh.

The following details of the Afrikander trials are given by Franklin White. The average ore assay was 8·28 dwt., the average residue assay was 1·4 dwt., and the theoretical extraction was 83·13 %. This is shown in greater detail in the annexed tables:—

A. OXIDISED ORE CRUSHED TO ⅛ IN.

| Mesh. | Weight. | Assay. | Dwt. in 100 t. | Residue. | Dwt. in 100 t. | Extraction. |
|---|---|---|---|---|---|---|
| in. | % | dwt. | | dwt. | | % |
| + $\frac{1}{20}$ | 34·8 | 5·7 | 198·36 | 1·7 | 58·65 | 70·4 |
| + $\frac{1}{60}$ | 35·1 | 7·3 | 256·23 | 1·2 | 46·20 | 81·9 |
| − $\frac{1}{60}$ | 30·1 | 15·0 | 466·55 | ·9 | 24·30 | 94·8 |
| | | 9·21 | 921·14 | 1·29 | 129·15 | 86·0 |

B. PYRITIC ORE CRUSHED TO 400 MESH.

| Mesh. | Weight. | Assay. | Dwt. in 100 t. | Residue. | Dwt in 100 t. | Extraction. |
|---|---|---|---|---|---|---|
| | % | dwt. | | dwt. | | % |
| Coarse + $\frac{1}{24}$ in. . . | 5·01 | 2·500 | 12·525 | 1·416 | 6·403 | 48·87 |
| Sands . . . . . | 74·81 | 3·833 | 286·823 | ·291 | 22·015 | 92·35 |
| Slimes . . . . | 16·00 | 3·208 | 51·328 | ·125 | 2·057 | 95·98 |
| Fine concentrates . | 1·92 | 90·000 | 172·800 | 10·500 | 14·700 | 92·00 |
| Coarse ,, . | 2·26 | 83·000 | 187·580 | 17·000 | 36·040 | 80·70 |
| | | | 711·056 | | | |
| Add free gold . . | .. | .. | 165·650 | | | |
| | | 8·76 | 876·706 | ·812 | 81·216 | 90·80 |

NOTE.—Treatment lasted 18¾ days, but was probably complete in 14 days. Ore only 3 ft. deep in tank.

C. PYRITIC ORE CRUSHED TO 300 MESH.

| Mesh. | Weight. | Assay. | Dwt. in 100 t. | Residue. | Dwt. in 100 t. | Extraction |
|---|---|---|---|---|---|---|
| in. | % | dwt. | | dwt. | | % |
| + $\frac{1}{20}$ | 2·4 | 4·0 | 9·60 | 1·25 | 3·625 | 62·26 |
| + $\frac{1}{30}$ | 20·9 | 3·5 | 73·15 | 1·75 | 40·775 | 44·30 |
| + $\frac{1}{60}$ | 37·2 | 6·5 | 241·80 | 1·70 | 58·140 | 75·96 |
| + $\frac{1}{90}$ | 12·8 | 8·5 | 108·80 | 1·20 | 14·880 | 80·33 |
| − $\frac{1}{90}$ | 26·7 | 9·2 | 245·64 | 1·00 | 27·200 | 92·33 |
| | | 6·789 | 678·99 | 1·44 | 144·620 | 77·00 |

NOTE.—Treatment 14 days.

These are highly instructive, particularly in showing how the extraction goes up when the ore is crushed fine, and even pointing to the advisability of re-crushing coarse particles instead of wasting cyanide on them for such poor results.

The cyaniding costs per ton are given by White as follows :—

| | d. | | d. |
|---|---|---|---|
| White labour | 11·18 | Lime (2·55 lb.) | 1·15 |
| Native ,, | 7·20 | Light and power | 2·30 |
| Maintenance | 3·45 | Stables | 0·85 |
| Assay charges | 3·60 | Coal, coke and sundries | 1·00 |
| Cyanide (·40 lb.) | 7·70 | | |
| Zinc (·31 lb.) | 1·08 | Total | 3*s.* 3·51*d.* |

Sometimes the consumption of cyanide ran as low as ·33 and even ·24 lb. per ton.

At the Lisbon-Berlyn mill, in 1899, the cyanidation of 27,457 t. of dry-crushed ore cost 3*s.* 6*d.* per ton, and gave an extraction of 6·32 dwt., equivalent to 68·3 %.

The cyaniding of dry-crushed ore is generally conceded to present some advantages in extraction, because the drying process renders the ore more porous by the act of dehydration, and in this condition it is more readily amenable to the influence of a leaching solution; but unless the drying process is under such control that it stops at dehydration, excessive heat may easily bring about an actual fusion, whereby the gold may be localised in small pellets which will indefinitely resist dissolution in weak cyanide liquors; or it may alter the constitution of the ore, and result in a cement-like mass which will set hard. Again, contamination of the ore by charcoal from the wood used in drying or roasting may be highly detrimental.

In New Zealand, in Utah, in the Black Hills, in Brazil, and in W. Australia, dry milling, even with argillaceous ores, has become well established, and there are few or no instances where, when the ore is crushed by suitable means, percolation cannot be applied, with or without suction, to the bulk of the product, while the fines can be eliminated for separate treatment. Nevertheless, in New Zealand itself, where dry milling for direct cyanidation first became an accepted practice, great efforts are being made to replace it by wet, notwithstanding the slimes difficulty.

In direct cyaniding on wet-crushed ore, the duty of the stamps is rendered extremely low (1¼ to 1½ as against 4 to 5 t.), by the fact that, when using cyanide solution instead of plain water, the volume has to be kept down to the minimum

which will permit an issue of the pulp, or to $1\frac{1}{2}$ to 2 t. instead of 9 to 10 t. of water per ton of ore milled. But Merricks confidently asserts that by wet crushing the Ohinemuri (New Zealand) ores the duty can be brought up to over 3 t., though he does not imply that this would be the case under the hap-hazard conditions of ordinary New Zealand practice. He bases his calculations on thoroughly modern methods and appliances, and supports his argument by quoting extensive experiments made at the Crown mill, where "the output has, with a very small and limited amount of water, been increased to $2\frac{1}{4}$ t. per stamp per day," provision being made for separate treatment of the slimes. This is all the evidence adduced.

On the other hand, Hooper * refers to trials made at the neighbouring Talisman mill, where "the ore was crushed wet, through 30-mesh screens, a ·85 % KCy solution being used in the mortar-boxes. Two vats were charged at the same time with ore to a depth of about 20 in. After siphoning-off the first solution, a 1·1 % KCy solution was used in vat No. 1, and 1·2 % in vat No. 2. After allowing these solutions to percolate for 36 hours, the ore was treated with sump and water washes, the time taken to complete the treatment being 6 days. The extraction results were as follows :—

| | Assay before Treatment. | | Assay after Treatment. | |
|---|---|---|---|---|
| | oz. | dwt. | oz. | dwt |
| Gold . . . . . | 0 | 14·16 | 0 | 2·15 |
| Silver . . . . . | 6 | 18·53 | 4 | 17 01 |

The total value extracted being 69 %. The slimes (7 % of the ore) which accumulated from the siphoned solution, assayed as follows per ton :—

| | oz. | dwt. |
|---|---|---|
| Gold . . . . . . | 0 | 7·29 |
| Silver . . . . . . | 7 | 7·32 |

In justice, however, to the method, it must be pointed out that the gold extraction amounts to 90 %, a by no means discreditable figure. Moreover, the low silver extraction may be due to external conditions, as for instance the prevalence of

* Trans. Inst. M. & M., vii. 54 (1899).

silver sulphide, which, as has been already remarked, dissolves with exceeding slowness in cyanide liquors.

An additional consumption of cyanide would seem to be inseparable from wet crushing with cyanide solutions, because of the increased bulk, circulation, and exposure to decomposing influences; though it is hardly likely that the iron of the mortar-boxes would have any calculable effect, as Hooper supposes, nor would there be any metallic mercury for the cyanide to appreciably attack (Hooper), because there is no occasion for using mercury in the boxes. According to McConnell, the cyanide consumption is the same whether wet or dry crushing is adopted, and Hooper says that in wet-crushing Talisman ore the loss amounted to "nearly 3 lb. KCy" per (long) ton of ore treated." Tailings from pan amalgamation are being successfully cyanided both in America and in India, and they contain much more mercury than any battery pulp could possibly do.

Experiments on direct cyaniding of wet crushed New Zealand ores are described at length by McConnell,* who got a duty of 1·4 to 1·68 t. per stamp per 24 hours, through 30 mesh, with 1·68 to 2·24 t. of water (solution) per ton of ore. The pulp, on leaving the mortar box, is passed over amalgamated plates to catch any coarse gold, and is conveyed by means of a launder into revolving distributors which discharge the ore evenly into the vats. As the pulp is charged into each vat, the ore settles, and the surplus solution, which contains more or less slimes, is allowed to overflow; it passes through two or more settling-vats for the purpose of settling the slimes, and then through the zinc precipitators, and is pumped back again to the storage vats, where it is made up to the required strength, and used over again.

When sufficient ore has been charged into the vat, it is allowed to settle, which operation is hastened by the addition of a little lime, and occupies ½ to 1½ hour. The clear solution is then run off from the top to within 2 in. of the surface of the ore, and a vacuum is applied to draw off the remaining solution through the ore. The washings are applied and

* Trans. Inst. M. & M., vii. 29 (1899).

drawn off in the usual way. It is advantageous to add a little lime to the stamper-box while the ore is being crushed; this helps the settling of the ore and slimes in the vat, and allows the overflowing solution into the settling-vats to go off comparatively clear. The mesh best suited to any ore can only be determined by actual trial, but he got, with average stone, the following results :—

| | 30 mesh. | 40 mesh. | 60 mesh. | 80 mesh. | 100 mesh. |
|---|---|---|---|---|---|
| Passed 100 mesh . . . | 69 % | 73 % | 68 % | 94 % | 100 % |
| Duty per stamp . . . | 1·65 t. | 1·69 t. | 1·18 t. | 1·28 t. | ·42 t. |
| Solution per ton . . . | 1·8 t. | 2·24 t. | 3·13 t. | 5·32 t. | 8·96 t. |

Some of the New Zealand mills have recently been converted from dry to wet methods, and others are following suit.

Among these is the Talisman, and it is interesting to read in the official report for 1899 that the "substitution of wet crushing will obviate expenditure on fuel for drying the ore, while the outputting capacity of the stamps will be at least doubled," and that "experiments have unquestionably proved that the ore is amenable to wet treatment, and that the wet process should be adopted in any new works erected." The following scheme of treatment is indicated :—(1) Passage of wet-crushed pulp over amalgamated copper plates for recovery of the coarser bullion (which would materially reduce the amount of cyanide royalty); (2) Separation of slimes by means of pointed boxes; (3) Treatment of slimes by cyanide process with mechanical agitation; (4) Treatment of sands (free from slimes) by cyanide percolation process. "It is possible that it may be found desirable to concentrate the ore (after separation of slimes) on vanners of Frue type, the concentrates thus obtained being separately treated with strong KCy solution, and the tailings (constituting the bulk of the ore) with comparatively weak solution."

**Leaching Plant.** *Accommodation.*—In laying out a leaching plant, provision must be made for giving separate accommodation to each day's output of ore or tailings to be treated.

For instance, Monday's product may occupy one or more vats or tanks, but Tuesday's must not mingle with it. The number of days' yield to be provided for depends on the duration of the treatment of each day's charge. Thus, a plant to treat 100 t. a day must have one 100-t. (or two 50-t.) leaching vats for every day that the charge is under treatment, with an additional vat to provide for such contingencies as repairs to vat or filter. If experimental work on the ore has shown that it will be necessary to leach and wash for 2½ days, and that ½ day is occupied in charging and discharging, it is evident that each 100-t. vat can be freshly charged once in 3 days. With the reserve vat, there will therefore be 4 100-t. or 7 50-t. vats for a 100-t. plant and a 3 days' treatment. Plants for other outputs and other periods are calculated on the same principles.

As to vat capacity, the needs of different ores vary widely, though it is commonly estimated that 30 cub. ft. of space per ton of ore is about an average. But concentrates require less room than this, while some argillaceous ores and tailings demand more, and the only safe guide is an actual computation made on the ore to be treated. In doing this, the question of moisture in the ore must not be overlooked. Ore coming from a wet-crushing mill may carry 10 to 20 % water, and the tonnage of such ore must be increased to that extent in charging the vats, or allowance must be made for lessened quantity in reckoning output, KCy solutions, etc. Assays being always made on thoroughly dry samples, great discrepancies in calculation of contents, extraction, and loss will occur unless the moisture be accurately determined and allowed for. Besides the actual space occupied by the ore or tailings, an additional depth of 6 to 12 in. is provided to allow of filter or false bottom below the ore, and of surplus liquor above it. For example, a circular vat 19 ft. 7 in. diam. by 3 ft. deep, affords 900 cub. ft.; to accommodate a 30-t. charge (30 t. × 30 cub. ft. per ton = 900 cub. ft.), this would be made, say, 3 ft. 9 in. deep, thus furnishing space for filter bottom and supernatant solution. A table of tank capacities is given at end of volume.

*Vat Dimensions.*—In the matter of dimensions there is considerable divergence in practice. Nothing is gained by making a vat larger than sufficient for a single day's supply of ore or tailings ; but, subject to that limitation, large size presents advantages in lessened cost of construction per cub. ft. of space provided or per ton treated, both in the vat itself and in the various pipe connections fitted to it ; it also affords some convenience and economy in working. On the other hand, large vats and their heavy loads necessitate more than ordinary care in securing the foundations against possible subsidence, in selection of material for construction, and in prevention of leakage. While the question of diameter is controlled only by mechanical considerations involved in the methods of charging and discharging to be employed, the question of depth has a most material bearing on the efficiency of the leaching. In extra depth lies the least costly way of securing additional cubic contents, and therefore it offers inducements for its adoption ; but experience has shown that the best methods of treatment are secured by comparatively shallow vats. According to Alfred James,* it is found that in all vats, except those of comparatively little depth, the lowest layer of residues almost invariably contains more gold than the general average of the contents of the vat. This is so marked, when the tailings contain clay, that it was thought at first that the solutions were decomposed, and the gold was deposited by it, as lumps were found to be actually richer after treatment than before they were charged into the vats. This is now considered to be due to the comparatively dry clay absorbing the cyanide solution, and to the increased pressure on the lower layers rendering permeation by the solutions more difficult ; it has been found almost impracticable to thoroughly wash these lower layers, even with a vacuum. Hence very few leaching vats are to be found which exceed 10 ft. in depth. In one recorded instance at the Langlaagte, 14 ft. is mentioned in 1895, but in 1898 it is given as 10 ft. Standard practice on the Rand is represented by the Henry Nourse plant.

* Trans. Inst. M. & M., iii. 375 (1895).

The following dimensions, casually noted, may be quoted as of interest in exemplifying recent practice :—

LEACHING-VAT DIMENSIONS.

| | Diameter, ft. | Depth, ft. | Capacity, t |
|---|---|---|---|
| Montana : Drumlummon | 38 | 9 | 400 |
| Utah : Golden Gate, Mercur | 50 × 25 | 5 | — |
| do. Del Mar, Mercur | $12\frac{2}{3}$ | 3 | 14 |
| Dakota : Black Hills | 24 | 3 | — |
| Transvaal : usual | 20 to 40 | $6\frac{1}{2}$ to 8 | — |
| do. Crown Reef (concrete) | 40 × 34 | 10 | — |
| do. do. | 31 | 10 | — |
| do. Henry Nourse, upper | 40 | 8 | 400 |
| do. do. lower | 37 | 10 | 400 |
| do. Langlaagte (concrete) | 40 | 10 | 400 |
| do. Robinson | 30 | 7 | — |
| do. do. | 40 | 7 | — |
| Victoria : S. German, Maldon | 25 | 9 | 110 |
| New Zealand : Waihi Silverton | 16 | 4 | .. |
| do. Moanataiari | 20 | 7 | .. |

*Vats, ranging.*—Leaching vats, or "percolators," as they are also called, are always arranged in rows in such a manner as to give the maximum facility for charging and discharging, this being the main source of expense in operating. The circulation of leaching-liquors and wash-waters is easily controlled, and can be made subsidiary to the handling of the solids.

*Vats, shape.*—In shape, leaching vessels are of two varieties, circular (then properly called "vats") and rectangular (when they should strictly be termed "tanks"). But in general usage the two names are employed without distinction, and they may now be regarded practically as synonyms, though Bosqui has adopted the former term for vessels containing solids and the latter for those holding liquids only. This is quite arbitrary ; and it seems likely to create more

confusion than if, at each mention of vats or tanks, their identity is specified where it is not obvious from the context. The circular shape is by far the more general, as it gives greater strength and resistance to wear; but the rectangular form may be adopted when the requisite skilled labour is not available for building the former.

*Vats, materials.*—The materials used in the construction of leaching vats are wood, iron or steel, and brick or concrete.

All the early plants adopted wood, and it is still much favoured in small installations, though wooden vessels demand much skill in erection, suffer deterioration leading to leakages under the influence of climatic changes and alternations of wetness and dryness arising from irregular operations, run risk from fire when idle, and absorb appreciable quantities of cyanide and gold. On this last point there is some divergence of opinion. An editorial comment * says that "the amount of gold required to saturate the apparatus of a new cyanide works, the vats if of wood," etc., "is apt to be astonishing," adding that the "gold thus absorbed is locked up indefinitely," and that while "part of it may be at some time recovered, part probably it may never pay to recover, even if the works were to be dismantled." But from some direct experiments made at Bodie by F. L. Bosqui,† "the indications are that in no case would absorption be much of a factor in gold losses, even in a vat of unprotected surface. Where leaching vats are thoroughly coated with asphaltum or paraffin paint, the losses are probably quite insignificant. The kauri pine used in New Zealand for cyaniding vats, and the California cedar and Oregon pine used elsewhere, may possibly, in some unexplained way, absorb considerable metal from solution, but it seems hardly probable."

Iron, and better steel, has come greatly into favour, being easily protected from attack by KCy solution, not absorbent, durable, and light.

Masonry has but few advocates, because, though absolutely non-absorbent, and innocuous to cyanide liquors,

* En. & Min. Jl., Aug. 28, 1897.

† *Ibid.*, Feb. 28, 1898.

detection of leakage is virtually impossible, and the least insecurity of foundation from any cause is quite fatal; nevertheless some of the largest S. African works have taken the risk; and the new Waihi (N.Z.) mill has concrete tanks of 200 t. capacity.

*Vats, wooden.*—All woods are not equally suitable for making vats. In S. Africa, the lumber most commonly employed, because it is ordinarily the only kind available, is Oregon pine; but it is soft and absorbent, and is responsible for a considerable loss of cyanide until it becomes saturated, amounting it is said to as much as 1 lb. per ton of its capacity. New Zealand is fortunate in affording a much better adapted wood in kauri, and the Australian colonies all possess their several kinds of hardwood (eucalyptus). American builders give preference to the Californian redwood ("cedar") or to Gulf cypress. The desideratum is a close and even-grained wood, free from knots and other blemishes, and "easy to work," that is yielding readily to the tool, so that there is no difficulty or irregularity in making joints.

The rectangular tank which often does duty at the commencement of operations may even be made of well-seasoned red or white pine "deals," $9 \times 3$ in., as described by A. James.* The joints are tongued and grooved; no packing of any kind is permitted, but a dressing of white-lead may be used. The bottoms are tied together by $\frac{7}{8}$-in. bolts running through the deals at every 13 in. or less, and the sides are let into the bottom, which is carefully gained $\frac{3}{4}$ in. for this purpose. The bottom projects 6 in. beyond the sides at the ends of the deals, but only $4\frac{1}{2}$ in. at the sides of the deals, otherwise the 3 in. gain would reach exactly to the edge of the first plank where it makes a joint with the second. The side and end bolts are $\frac{3}{4}$ in. diam., running through the sides and bolting them to the bottom, and are at equi-distant spaces with the bottom transverse bolts. The bolts at the corners of the sides are as close as possible, and are connected by a triangular iron cap to keep the corners well down. The projecting ends of the sides of the tanks are drawn together

* Trans. Inst. M. & M., iii. 378 (1895).

by 3 $\frac{3}{4}$-in. bolts to keep them tight on to the ends. At every 30 in. or less, struts $4\frac{1}{2} \times 3$ in. and the height of the tanks are mortised into the bottom and fixed against the outer side by coach screws, to afford additional rigidity. Three $9 \times 3$-in. planks run across the top of the tanks at equal spaces, and act as distance-pieces as well as tie-rods for two of the sides, whilst the other two sides are supported by distance-pieces between them and the adjacent tanks. The vertical side bolts should be continued through the $9 \times 6$-in. foundation-pieces, to afford additional rigidity to the bottom. These tanks are of any size from $11 \times 9 \times 3\frac{3}{4}$ ft. to $24 \times 16 \times 5$ ft. All the first working plants in Africa and New Zealand were constructed of this shape.

A very modest plant erected lately in New Zealand, to treat a small accumulation of slimy tailings from pan amalgamation, embraced wooden tanks 2 ft. deep and 12 ft. sq., of $1\frac{1}{2}$-in. lumber, well nailed together, and braced on the outside at every 4 ft. Over each joint was laid a strip of canvas 3 in. wide, a coat of hot asphalt being first applied. A sheet of medium canvas 16 ft. sq., sewed with double seams, was laid down in the vat and folded up in the corners, the side next the board being covered with a good layer of hot asphalt just as it was laid, causing it to stick fast to the wood throughout. The upper edges were tacked to the vat, no other tacks being put through the canvas. The inside of the canvas lining was then thoroughly asphalted, with special care about the corners. The vat floor inclined $1\frac{1}{2}$ in. in 12 ft., both east and north. In the north-east corner, 1 ft. from sides, the floor had been cut $\frac{3}{4}$ in. deep in a saucer-shaped depression, and a hole for a $1\frac{1}{4}$-in. pipe bored. This was put in through a hole in the canvas, and bolted tight in hot asphalt, with 3-in. washers on the canvas and on the plank below.

Circular vats are made of well-seasoned staves, $4\frac{1}{2}$ to 5 in. wide and 3 in. thick for vats up to 25 ft. diam., and 5 to 6 in. wide and 3 to 4 in. thick for vats above this size. Each stave is shaped to the radius of the size of the tank—not quite truly, but slightly feather-edged inside, to allow for the swelling of

the skin of the timber in contact with liquids, which thus makes the joint perfectly tight. No tongues and grooves, or dowels, or bruised joints are allowed in making the stave joints, which must be planed perfectly true and accurately fitted. The staves are at least 9 in. longer than the inside depth of the tank, and this allows for a 3 in. gain $\frac{3}{4}$ in. deep to receive the bottom of the tank, and for a projecting horn of 6 in. The sides are vertical, and the bottom is made of 12 × 3-in. planks, planed to make true joints ; tongues and grooves, bruised joints, or packing of any kind are not allowed, but white-lead may be used, and the bottoms are bolted together by $\frac{7}{8}$-in. bolts running through the planks. In vats not exceeding 8 ft. diam., the bottoms are dowelled, and bolts are dispensed with. The sides are held together and to the bottoms by round iron hoops, one for every foot of height of the tank, and one extra at the bottom ; for a 25-ft. tank, the two lowest hoops are 1 in. diam. and the others are $\frac{7}{8}$ in. They are made in about 20 ft. lengths, each length having both ends threaded and connected to the next by tightening bolts or lugs. This enables slackness in any particular part to be at once taken up by tightening the adjacent nuts, and is also of advantage in fitting the hoops to the vat.

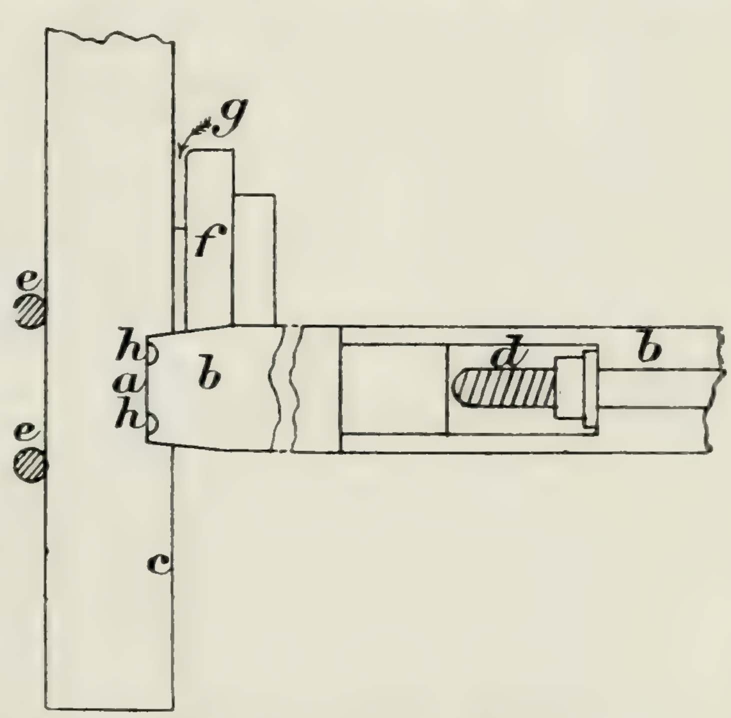


Fig. 141.—Tank Bottom.

The details of construction are shown in Fig. 141 : *a*, gain in stave, 3 × $\frac{3}{4}$ in. ; *b*, bottom ; *c*, horn or chime ; *d*, bolt ; *e*, iron hoops ; *f*, wooden ring and support, behind which, in space *g*, the edge of the filter-cloth and the caulking-rope are wedged ; *h*, grooves in which round rubber packing is sometimes inserted as a precaution against leakage.

Wooden tanks of all kinds need to be dressed quite smooth inside, and should receive a preservative coat both

inside and out. For the outside, gas-tar or lead-paint is cheap and serviceable. For the inside, preference is given by most authorities to treatment with paraffin, either applied dry and rubbed in or brushed on as a varnish. Mactear* speaks highly of two other kinds of dressing:—(*a*) prepared by melting ordinary rosin and stirring into it chips or cuttings of waste vulcanised rubber, and (*b*) prepared from alumina soap, which, when freed from water, fuses very easily, and is soluble in paraffin or turpentine, forming a very hard and tough varnish. No information is available, however, concerning the possible action of these compounds on cyanide solutions. There would seem to be some risk about (*a*) on account of the sulphur in the vulcanised rubber. The author has used common gas-tar, applied hot, without any evil results; and Bosqui recommends equal parts of coal-tar and asphalt as being equal to and cheaper than paraffin-paint.

*Vats, steel.*—Against steel vats it has been urged that leaks are more difficult to remedy and repair than when they occur in wooden ones. This is open to considerable doubt, and it is quite certain that leaks are much more easy of absolute prevention in sheet steel than in wood. The great difficulty with steel vats is to obtain an accurately level bottom, because the effect of riveting is to spread the metal, so that a certain amount of buckling is practically unavoidable. But no material drawback arises from it.

The thickness of plate commonly used is $\frac{1}{4}$ in., which, in tanks of ordinary dimensions, is much in excess of the actual needs so far as tensile strength is concerned; but in choosing a thinner metal, there is greater difficulty in securing a tight joint, and more risk of shearing by the rivets. Nevertheless the author has used $\frac{1}{8}$-in. sheets for agitation vats 12 ft. diam. × 7 ft. deep, and found them amply rigid and tight, using a rim of angle-iron 2 × $\frac{1}{4}$ in. at top and bottom.

It is economical to choose a depth of vat in exact accordance with the size of plates obtainable, the seams in the sides or periphery running vertically.

In some localities, engineering firms can be found to

* Trans. Inst. M. & M., iii. 411 (1895).

undertake the construction of steel tanks, which may be transported to the mine ready-made either wholly or in part, the sections packing conveniently within each other for conveyance by rail or road; but in other cases, the sheets must be imported, and the whole labour of erection be carried out on the spot. In this latter case, the instructions issued by the Cyanide Plant Supply Co., London, will be found very useful for guidance. They recommend that the bottom plates be first raised on the framework supporting the vats, packing being introduced to keep them about 2 ft. clear, thus allowing room for putting in and holding up the rivets which fasten the bottom sheets together. When the angle-iron curb or rim has been bolted on round the edge, and the whole of the bottom plates have been properly bolted together in position, the bottom is riveted up, the rivets being passed through from the outside. When completely riveted up, the bottom is lowered into its place on the platform. The side plates, with the upper angle-iron (and stiffening-piece, if there be one) are next bolted in position, and riveted up in the same fashion. Finally, the edges of the bottom angles and the cross laps of the plate seams are chipped, fullered or trimmed, and scraped quite clean. No caulking should be necessary, but the clean edges of the plates may have a coat of white lead; should a water test show leaks, then efficient caulking must be resorted to, though it is most undesirable. Riveting is done hot, and the heads are hammered over and not "snapped." The bolts used for holding the plates in position till riveted may be about 3 ft. apart.

The construction of the bottom joint is shown in detail in Fig. 142: *a*, side; *b*, bottom; *c*, angle-iron; *d*, rivet; *e* ring of $3 \times \frac{3}{8}$-in. flat iron, chamfered at *f*, for supporting filter-frame, and behind which the filter cloth is fastened by a caulking-rope. It is of course understood that a rivet at *c* unites the bottom to the angle-iron. The inner ring *e* is often omitted; and the angle-iron may be placed inside or outside the tank.

Iron or steel tanks should be covered with a coat of tar, paraffin, or other protective material, both as a pre-

servative and to prevent contact between metal and KCy solution.

*Vats, Masonry.*—Concrete or brick-lined pits need to be made of the very best materials and with most minute regard for perfect foundations, so that leakage may be impossible of occurrence, because it cannot be discovered and remedied.

*Vat filters.*—Filter-bottoms were originally constructed of iron-wire gauze, with a covering of cotton twill; the gauze was expensive, and was acted on by cyanide, and pebbles and coconut matting were substituted. Pebbles are apt to clog, and thus interfere with efficiency. Modern practice is to run

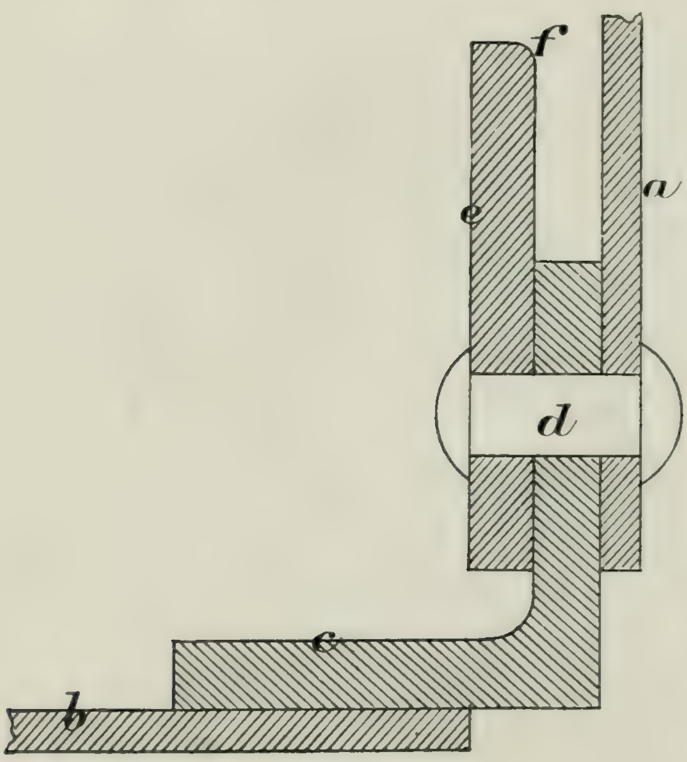



FIG. 142.—TANK BOTTOM.

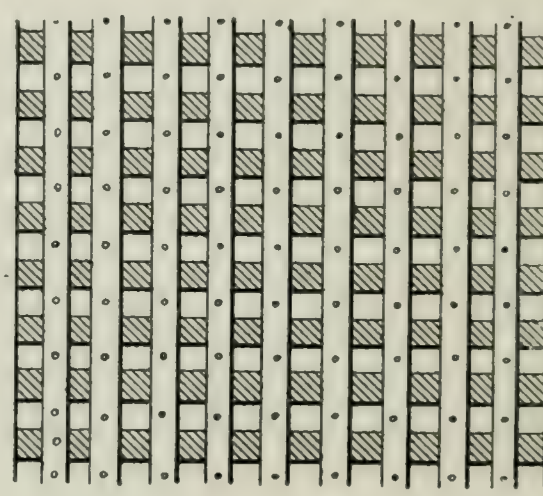


FIG. 143.—TANK FILTER.

a ring 3 in. high around the bottom of the vat ½ in. distant from the inside of the staves; and in the space between it and the staves, are tightly packed the ends of the filter-cloth and a caulking-rope. In wooden vats this ring is of 1½-in. wood, and is fastened down by screws or wooden pins into the bottom; in steel tanks, an extra angle-iron is riveted to the bottom. Within the circle formed by the ring, or, in the case of rectangular vats, within the parallelogram formed by the bounding slats, 2 × 2-in. filter-slats are laid parallel across the planks forming the bottom of the tank, at a distance of 6 in. from each other. In vats not exceeding 5 ft. in height,

these slats may be 10 in. from each other. At every 12 in. they are screwed or pinned to the bottom through wooden squares 2 × 2 × 1 in. high, which act as distance-pieces to keep the slats off the bottom, and allow ample room for the solutions to flow beneath. If the vat is more than 8 ft. high, the filter-slats must be of 3 × 2-in. timber, to stand the increased pressure. Across these slats, 1-in. semi-circular ordinary ceiling moulding is nailed in parallel lines, rounded side uppermost, at intervals of 2 in. from centre to centre—this gives 1-in. spaces. At every 8 in., that is just over the filter-slats, cross-pieces are put in, thus forming a frame-work with openings of about 7 × 1½ in., which gives an area of 10½ sq. in.; over this a filter-cloth of calico, cotton-twill, canvas, jute, or coconut matting is laid.

The open space seems large, but in practice the filter-cloth will stand a pressure of 15 lb. per. sq. in. perfectly well without breaking. This is a considerably higher pressure than would be obtained in a 14-ft. vat without vacuum suction. Fig. 143 gives an idea of the appearance of the filter-bed. The cost is much lessened by widening the spaces between the slats and filling the intervals with clean pebbles.

In steel tanks, the filter-bed may be made of 1 or 1½ in. angle-iron strips riveted to the bottom, the spaces being filled with coarse sand. The covering of matting and shovelling slats is laid on this.

Coconut matting or jute is usually employed for leaching unassisted by vacuum suction, otherwise cotton-twill or canvas is preferred. The actual filter is protected by a layer of coconut matting over it, and then by slats of wood arranged to prevent the shovels from cutting the cloth when the residues are discharged by manual labour.

*Vat doors.*—Where water is available, provision is made for sluicing away the residues through a door in the side of the tank, placed at such a level that the bottom edge of the opening of the door is about ½ in. below the level of the top of the filter. These doors are usually 12 × 6 in., though in some cases they are made of larger sizes, a plate with a rubber ring on its flange being pressed tightly by a screw on to a

frame, bolted on to the opening in the vats. The screw and thrust bar are made to come away from the lugs when the door is opened, so that no damage may be done to them by the rush of pulp. When the residues are discharged by hand, and the sides of the vat are over 4 ft. high, a door is built in

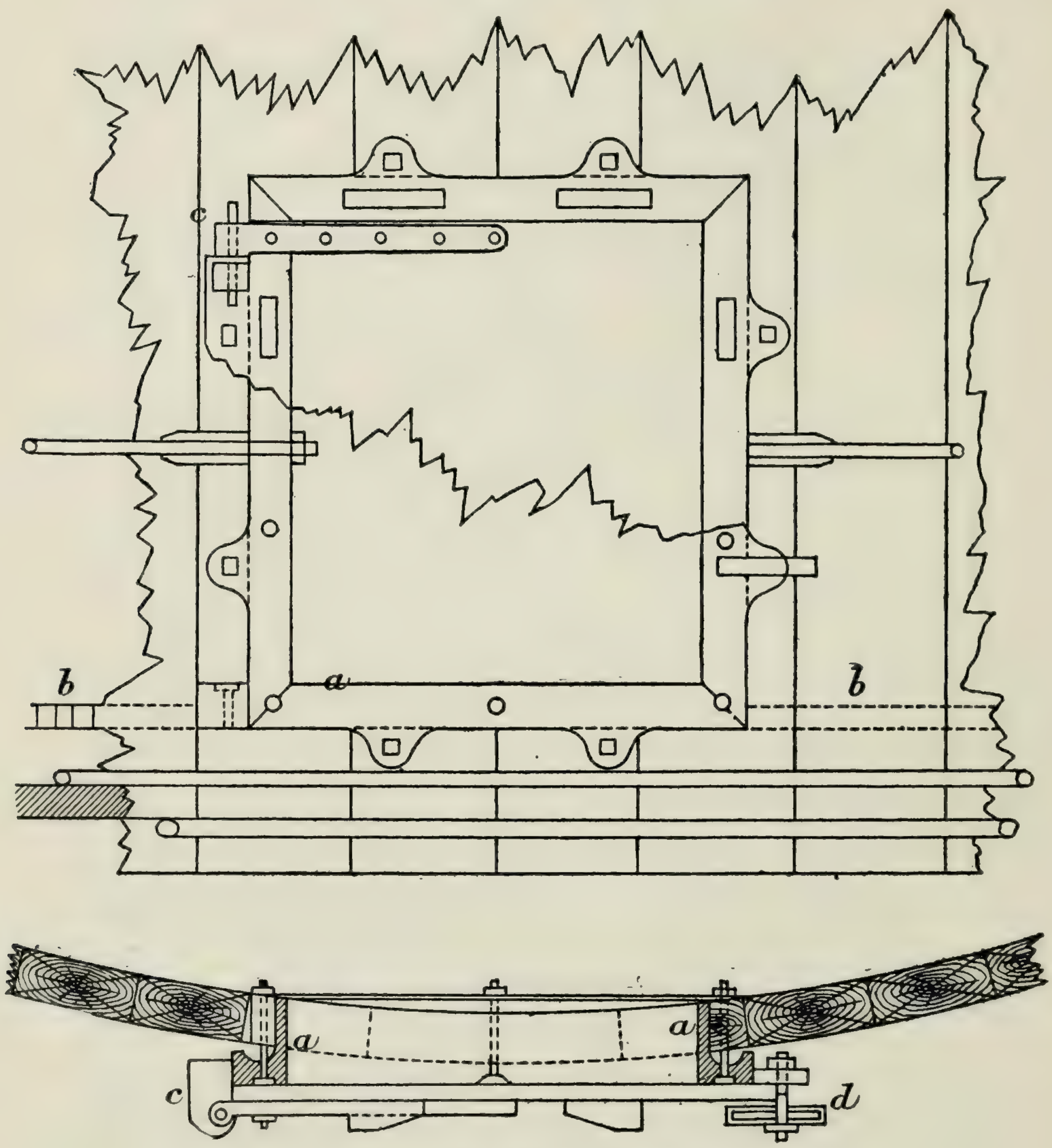



Fig. 144.—Tank Door.

the centre of the bottom, and the residues are shovelled into trucks below. Large tanks may have 6 or 8 of these doors. Where this bottom discharge is not convenient, doors have been placed in the sides at a suitable height, through which the residues are shovelled. In masonry vats, tram-lines are laid over the filter, and trucks are run into the vats through iron doors built into the sides.

Much variety exists in the patterns of doors. An early form of side-discharge door for wooden vats is Feldtmann's (Fig. 144), in which an iron frame *a* is built on to the vat, and securely held by through-bolts, the bottom of the opening just clearing the top of the filter *b*; the door is hinged at *c*, and, when shut, is held tightly against a rubber packing (such as old vanner-belt) by clamps *d*. Instead of hinges, the weight of the door may be borne by a small pulley and counterpoise. With a substantial frame, lugs projecting from it may receive

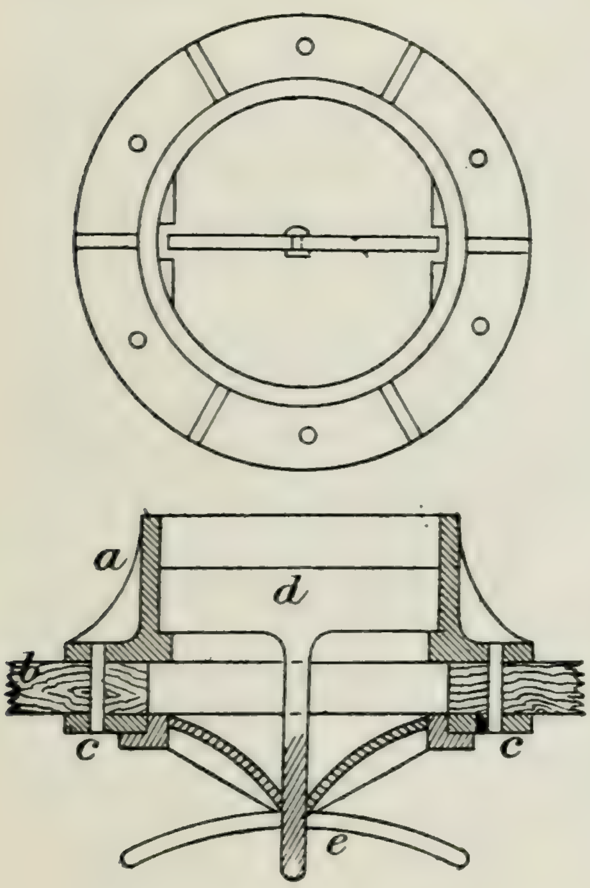



Fig. 145.—Tank Door.

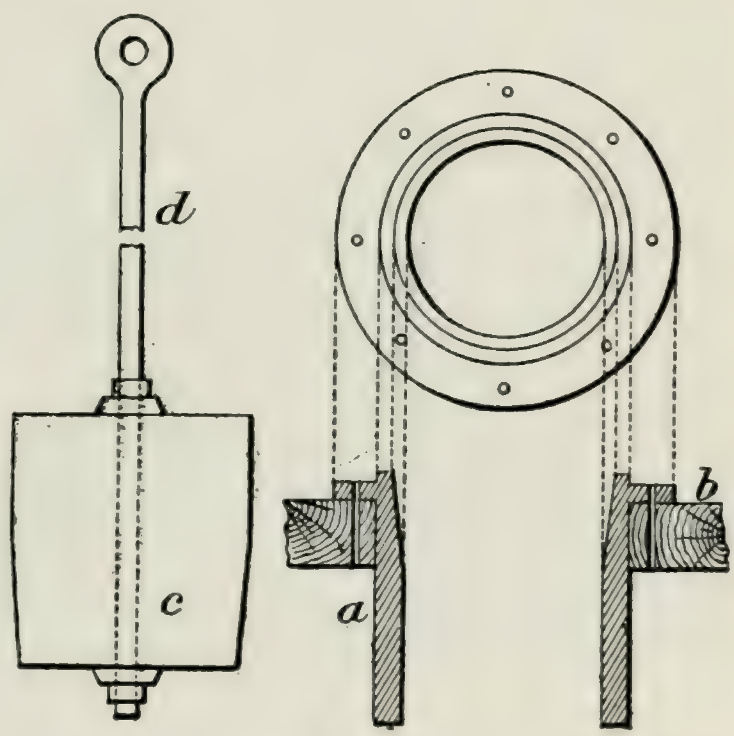



Fig. 146.—Tank Door.

hardwood wedges, as in McCone's door (Fig. 145); or a strong swing bar may be substituted; or half a dozen stout bolts carrying nuts. With iron tanks, the side may be stiffened by a rim of 2 × ½-in. flat iron, and the door be made to the same radius as the vat, ¼-in. plate sufficing. In all side-discharges there is no interference with the filter-bottom. It can be easily adapted to discharging either by sluicing or by hand. In the latter case, when the vat is large, the door is made capable of admitting a truck into the vat. In the former, a sheet-iron hood or apron can be contrived for guiding the issuing pulp into the launder.

Bottom discharge is preferred by most, whether for manual or hydraulic operation. The door in this instance will consist of a strong flanged collar, as at *a*, Fig. 145, which represents the Butters arrangement, projecting from the true bottom *b* of the vat to the full height of the filter-bottom, and fastened in the aperture of the vat bottom by bolts *c* through the planking. The plug *d* is held in place by the turnbuckle *e*. The door opens outwards. In Parr's door (Fig. 146), the collar *a* is similarly bolted to the vat bottom *b*, but it is prolonged downwards into the sluice to avoid splashing in hydraulic discharging, and the plug *c* is raised inwards by the rod and loop *d*, often requiring considerable force to dislodge it, owing to the wood swelling, but this may be remedied somewhat by dressing with paraffin paint; a clay luting is necessary before filling the vat. All these doors are much less satisfactory than the patent pressed-steel bottom-discharge door made by the Cyanide Plant Supply Co., which is being universally adopted. In discharging a deep tank, the residues issue more freely into the outlet if this is within a few feet of the surface; otherwise they have to be pushed down with long poles.

*Vat foundations.*—Foundations for large tanks are usually constructed of masonry, built in walls of the required height —say 5 ft. The outer walls are rounded to the shape of the vats, and, with the inner walls, are arranged to admit of a tram service under the discharge doors, as also for access to any part of the bottom of the vats to examine for leakage. Wooden vats do not rest directly upon the masonry walls, but upon intermediate bearers of 9 × 6-in. timber, running in a direction at right angles to the planks in the bottom of the vat. The bottom of the staves must be quite free from the foundations, and bear no strain; leakages being most apt to occur at the point where the bottom is gained into the staves, freedom of access must be provided for. Steel vats are mounted on floored platforms. In vats up to 25 ft. diam. and 6 ft. high, wooden foundations answer perfectly well. The vats are supported on 8 × 6-in. bearers, about 3 ft. apart from centre to centre, resting on posts 6 in. square, braced by

diagonal struts of $4\frac{1}{2} \times 3$-in. timber. The sills are $8 \times 6$-in. planks, and these again rest on mud-sills, which are intended to prevent any settling of the foundations. The dimensions above given are for 25-t. vats, with 2 ft. 6-in. posts; larger vats or higher frame-work requires stronger posts. The foundations are octagonal in shape, the radius of the octagon being at least 3 in. less than that of the internal diameter of the vats (to allow the ends of the staves to be quite free), when wooden vats are used. Fig. 147 illustrates both masonry and wooden foundations.

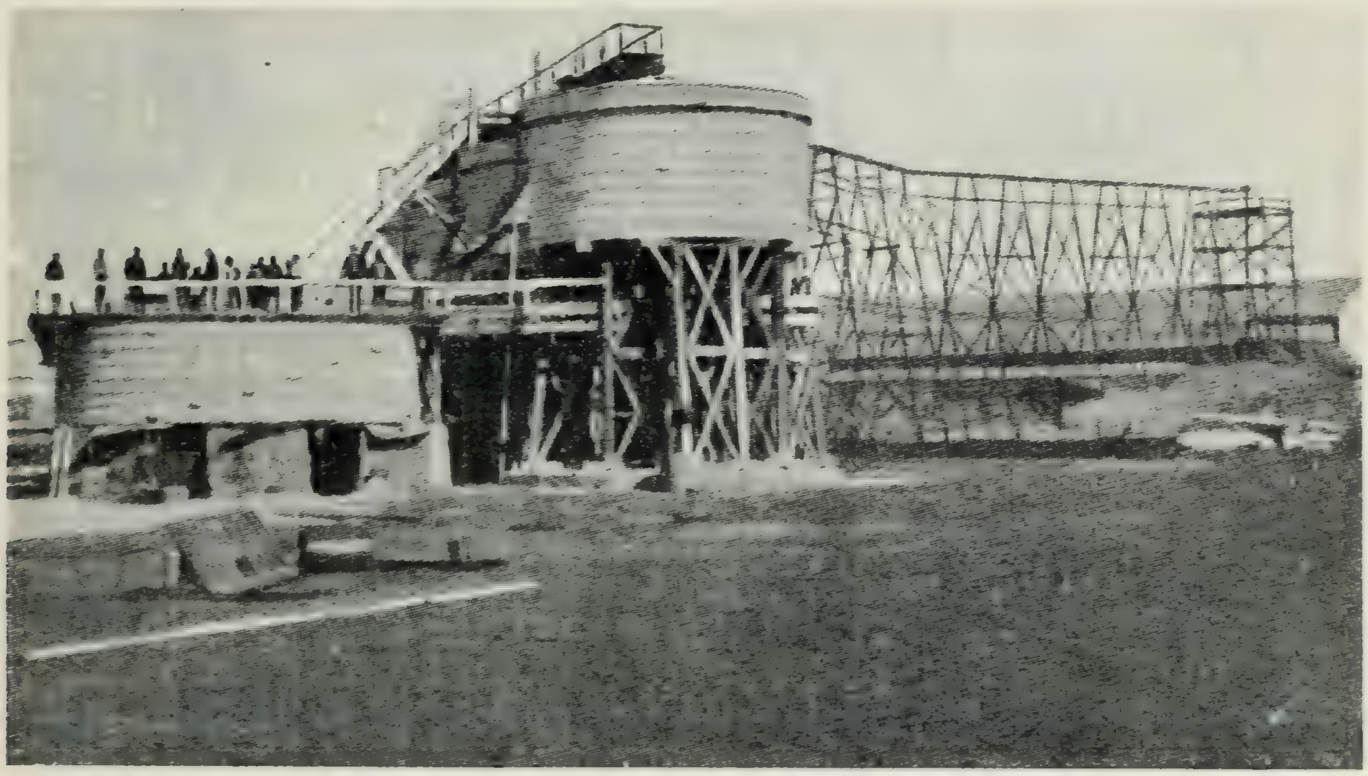


FIG. 147.—TANK FOUNDATIONS.

At the Henry Nourse mill, Transvaal, $\frac{1}{4}$-in. steel vats are carried on steel girders and steel tubular pillars.

Sometimes the posts of the upper vats stand within the lower vats, resting either on the bottom, or, preferably carried through to the lower-vat bearers.

Practice varies as to having the bearers exactly level. Some engineers insist on it, whilst others allow a slope of 2 in. on the entire length of the vat or tank towards the sluicing door or liquor exit, to assist in the discharge of tailings and solutions. If the bearers are made exactly level and sluicing is employed, the filter itself may be constructed

with a slope towards the discharge door, whilst channels are formed in the bottom of the vat to lead the liquor towards the outlet.

*Vacuum apparatus.*—To hasten the leaching of material through which KCy solutions and wash waters do not percolate freely, the effect of atmospheric pressure is availed of by producing a vacuum beneath the filter-bed of the leaching vat. The plant comprises two distinct parts, viz. the exhauster and a receiver for the exhausted liquor.

The exhauster may be a plunger vacuum-pump, or some form of ejector, or a natural suction may even be caused by prolonging the exit pipe down to the receiver when the latter can be placed at a much lower level.

Where power can be conveniently derived from neighbouring shafting at the necessary speed (about 150 strokes a minute) preference is generally given to a vacuum-pump, the 6-in. size requiring about 2½ h.p.

The ejector needs nothing but a jet of steam at moderate pressure, and is a simple contrivance which can be made on the spot. It depends simply upon the suction produced by throwing a jet of steam from a nozzle into a closed pipe. At the Simmer and Jack works, the issue pipe is merely connected with the engine exhaust.

To secure the best results and most economical application of the steam, certain proportions must be observed in the bore of the respective pipes. This is carefully attended to in the ejectors made by Meldrum Bros., the well-known furnace engineers. They furnish a variety of sizes, ranging from 1000 to 60,000 cub. ft. of air per hour, with steam inlet from ½ to 4 in., and costing 50*s.* to 17*l.*, or 90*s.* to 25*l.* when provided with a regulating spindle-valve for irregular work. They are made to operate at various steam pressures, but 50 lb. is usual.

Hydraulic power may be similarly used where water pressure is abundant.

The receiver is a boiler-shaped vessel, of iron or steel plate, varnished inside with paraffin to prevent contact of KCy solution and iron surface, fitted with vacuum and water

gauges, discharging into storage tanks, and connected by a 2-in. air pipe with the exhauster and by a 3-in. liquor pipe with the main line from the leaching tanks. A receiver 13 ft. long and 3 ft. 9 in. diam., holding about 4 t., has sufficient capacity for a 2000-t. plant, where strong liquors only are exhausted, but must be duplicated if weak liquors are to be similarly treated.

*Agitators.*—One of the first forms of agitator used was a revolving steel barrel, closely resembling a chlorination barrel

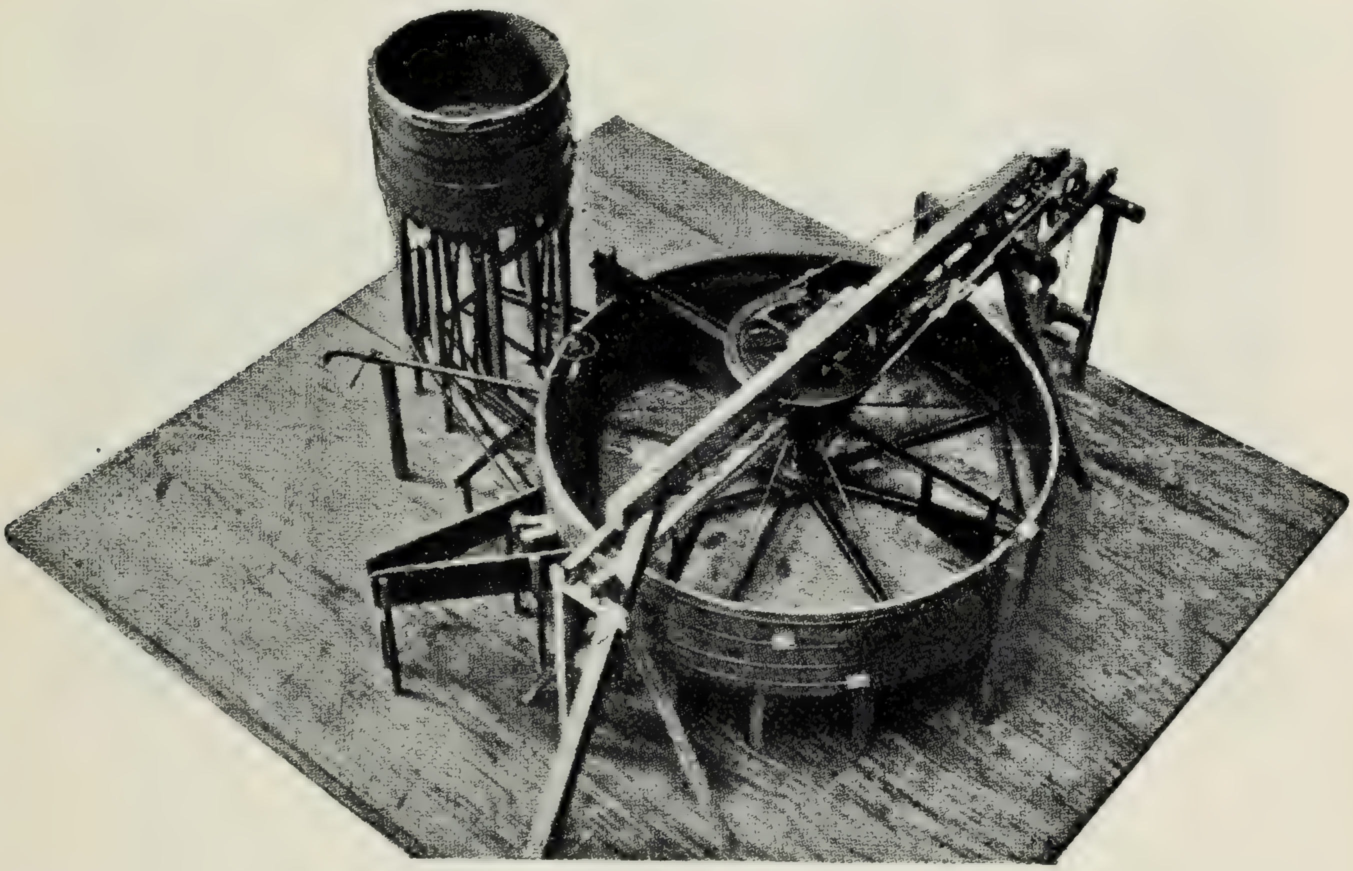


FIG. 148.—DEEBLE AGITATION PLANT.

(p. 433) but without lead lining. This was adopted at the Golden Reward works, in the Black Hills, in 1893, after many experiments, though nothing can be said in its favour, and it has probably long since been discarded.

A step in advance of this is the open barrel standing on end, or small vat, with an upright shaft carrying agitator blades. This idea has been very well carried out at several works in Australia, where it is often called Duncan's or Deebles' system. To the ordinary leaching tank is added mechanism for maintaining the pulp in a state of agitation, as seen in Fig. 148.

In a plant erected at Lucknow, N.S.W., by the author, the tanks are 12 ft. 6 in. diam. and 7 ft. deep (Fig. 149). An overhead framework carries a line shafting with cog-gear for

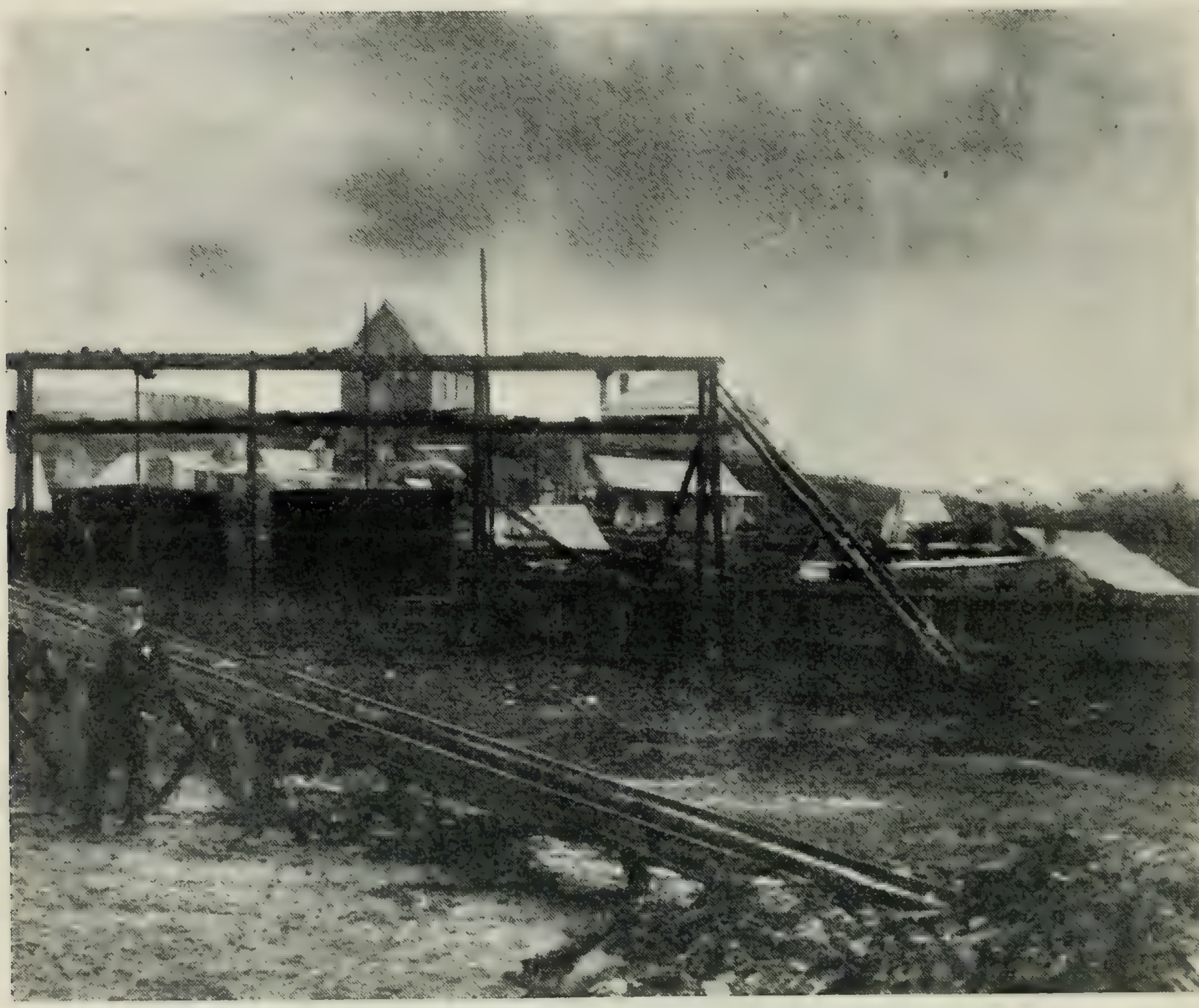


FIG. 149.—AGITATION TANKS.

each tank. A vertical spindle (discarded stamp-stems were used in this instance) 3 in. diam. in each tank carries 8 arms of 2 × ½-in. flat iron (old rails which had been pulled out of

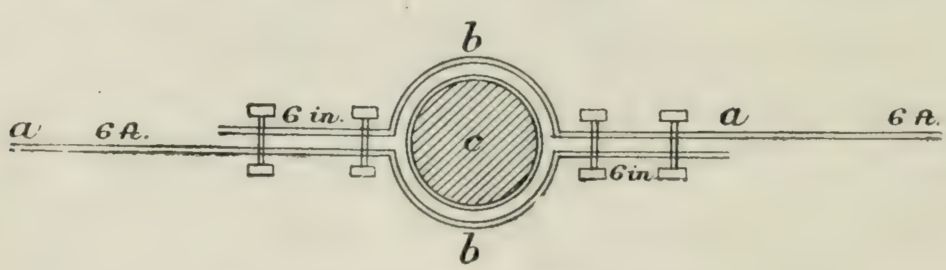



FIG. 150.—AGITATING ARMS.

the mine), one set of four being placed close to the bottom, just clearing the rivet heads, and the other about midway up. These arms are fastened to the spindle in the simplest manner, as shown in Fig. 150, two bars *a* 7 ft. long being bent at about

6 in. from their ends into a half circle *b* of slightly less radius than the spindle *c*, and clamped by a couple of ½-in. bolts on each side. The second pair of arms is placed at right angles to the first pair and as close to them as possible, forming the set. The gear has clutch and lever fittings for throwing it into and out of action, and the key-way is cut the full length

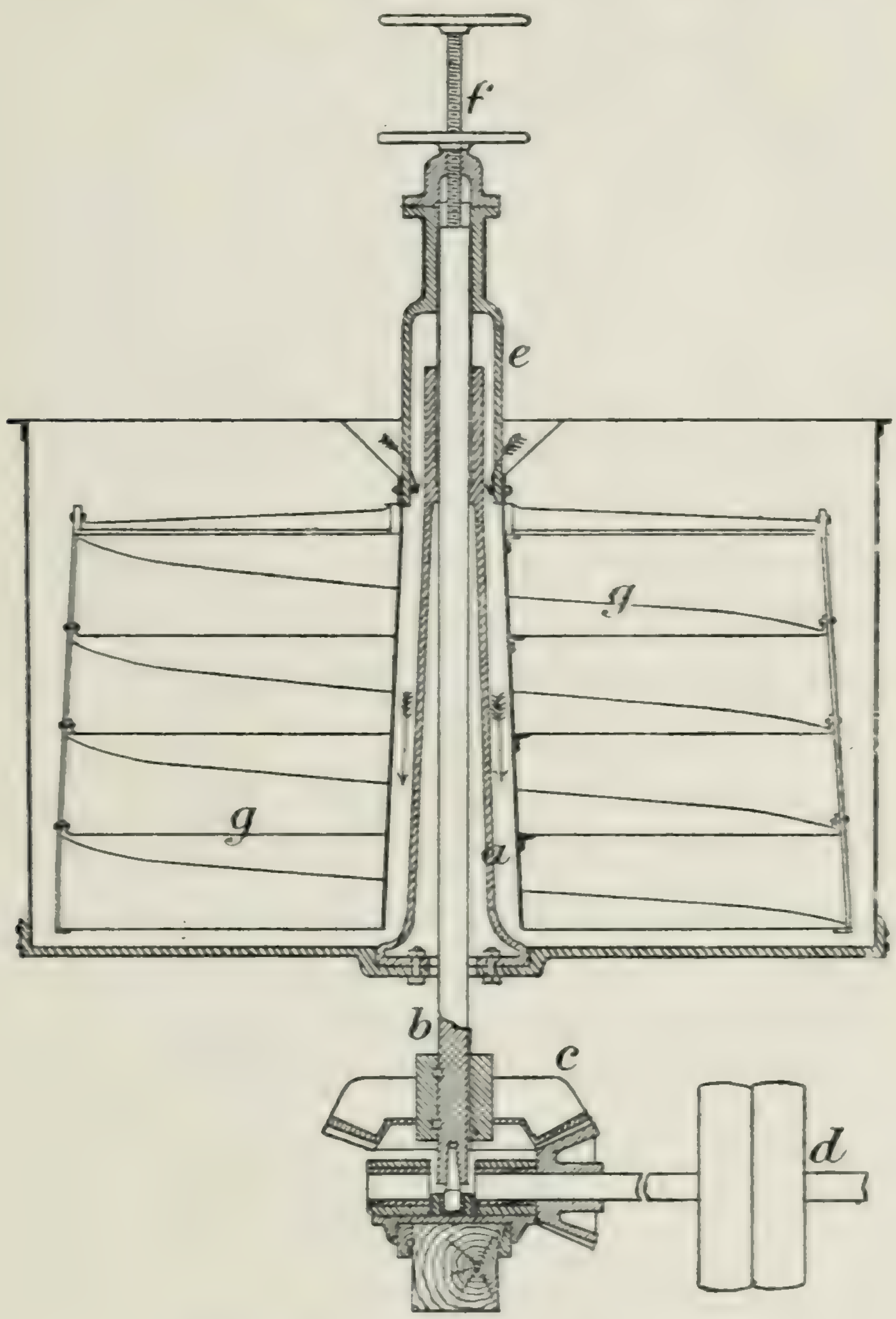



FIG. 151.—BONNEVIE AGITATOR.

of the spindle, so that the agitators can be raised and lowered while in motion. The power required is so small as not to merit consideration—less than 1 h.p. per tank, when charged with slimes containing 50 % moisture. A series of baffle-plates, one at each seam and riveted with it, assists the agitation.

The Bonnevie continuous leaching tank (Fig. 151),

measuring 8 ft. diam. × 5 ft. deep, consists of a cast-iron bottom with steel-plate sides. In the centre of this bottom is secured a cast-iron cone *a* ending in a guide-bearing for the vertical driving-shaft *b*, connected by bevel-gearing *c* with horizontal shaft *d*. It carries at its upper end an adjustable driver to which the agitator *e* is attached. By means of hand-wheel and screw *f*, the agitator is raised or lowered while in motion. The agitator *e* is a low-pitch screw conveyor, completely filling the diameter of the tank and presenting a surface having 4 times the area of the bottom, this form being selected as presenting a settling surface for the ore, so that, in case of a shut-down, the pulp settles on the blades instead of on the bottom, and is thus readily prevented from packing. The agitator blades *g* consist of steel sheets bolted to the driver, and stayed by angle-irons. The average speed is 15 rev. per min. It is claimed that 75 % pulp and only 25 % liquor can be efficiently agitated. The feed is automatic by funnel *h*, between the central cone and driver to the bottom, whence the stream rises, and overflows at a lip on the margin of the tank.

Another form of agitator introduced by W. Duncan at Charters Towers, Queensland, resembles a boiler divided longitudinally, and measures 14 ft. long, 5 ft. deep and 9 ft. 6 in. wide at top. It is of ¼-in. steel. A shaft runs longitudinally in the centre of the top and carries 25 "beaters," pieces of 3 by 2-in. wood, which are let into cast-iron sockets, clamped on to the shaft. These beaters are arranged screw fashion along the shaft, and less than half of them are engaging the slimes at one time, the beaters being screwed on to the shaft at different angles. The beaters are of such a length that when they revolve they extend to within ½ in. of the side and bottom of the vat, so there is no possibility of a sediment depositing, especially as the space between the beaters is only about 2 in. The charge is 10 t. of slime, 1000 gal. of KCy liquor having first been admitted. A series of taps allow of withdrawing the gold-cyanide liquor in a clear state.

The Cyanide Plant Supply Co., London, have worked out

all the details of the agitation process and amplified and improved the apparatus in many respects.

Their conical vats are made in any required size, very well built, and most conveniently equipped.

Each plant requires to be designed specially to suit the material treated. The size of some of the most recent vats is 40 ft. diam. by 14 ft. deep at the sides and 17 ft. deep at the centre. They are made of steel, usually $\frac{5}{16}$-in., except the conical plate at the bottom, which is $\frac{1}{2}$ in. thick.

The slime treatment tanks at the Crown Reef, Transvaal, are of sheet iron with concrete bottoms dished in the centre.

At Silver Star, Montana, the agitating vat is 16 ft. diam. and 8 ft. deep. The central shaft carries only two blades, opposite each other, and shaped like a screw propeller. At a speed of 18 rev. a minute they break up all concretions and effect thorough agitation. The charge is about 12 t. of slimes, with liquor ranging from 1 to $2\frac{1}{2}$ t. per ton. The cost of a single agitator complete was 60*l.*, but this would be reduced in a number. The size named receives all the slimes made by 5 stamps.

Sometimes use is made of a swiftly-revolving pipe 18 in. diam. fitted internally with helical curved or rifled vanes, which force the pulp against an iron plate on the bottom of the tank.

Centrifugal pumps are found to be very efficient agitators.

Park has employed a somewhat complicated arrangement which is supposed to possess advantages in celerity and continuity. One or more circular vats, 20 to 25 ft. diam., are connected with an air-pump and vacuum cylinder. Each vat has a false bottom covered with fine filter-cloth, and 4 rotating arms carrying rubber strips which brush the surface of the filter and keep it from choking with slimes. It is open to doubt whether, on a commercial working scale, these rubber scrapers would be so efficient as to ensure the necessary rapidity of filtration.

Old slimes require to be broken up in a special tank. This may contain a long narrow ribbed trommel revolving under water, as was successfully adopted by the author

at Lucknow. Or preference may be given to a contrivance thus described by Alfred James.—In a tank about 4 ft. deep and 5 ft. diam. is a "pierced internal cone, with a revolving screw or vane at the bottom." The pulp is drawn through the cone past the rapidly circulating vanes, and returned through the holes in the cone to the upper portion of the vat.

This prepares the pulp for the proper agitating tank.

**Charging and Discharging Vats.**—Various circumstances help to govern the method by which leaching tanks are to receive their charge of ore or tailings for treatment and to be relieved of the solid residues after treatment.

The charge may be in the wet state, as it comes from a stamp battery, or in a dry state as delivered by rolls and similar machines. It may be drawn from accumulations lying at a lower level than the tanks, or from sources situated above them. In all these cases, advantages will be found to lie in some particular mode of delivery.

*Delivery.*—For transportation of wet tailings of current production, nothing can be simpler than gravitation through open launders, set at the requisite grade of ½ to 1 in. per foot, even though it be necessary to first elevate them to a height affording the necessary fall. The cheapest of all transport is a stream of water.

Tramways and trucks are often used for old tailings and for dry-crushed ore. In that case, the tramway should in no part have a bearing upon the vats or their platform, as the vibration imparted by travelling trucks causes leakage of joints in wooden vats, and packing of the charge in all.

The conveyance of dry-crushed ore to the leaching vats is not a simple matter, on account of the dust. At Waihi, the worst possible practice is in vogue, viz. drawing the ore from bins through open shoots into ordinary trucks, which are pushed by men on stagings over the vats and dumped. The dust created is beyond description, and it seems wonderful that men can be got to do the work. In more advanced mills, pipe and belt conveyors are used, but the latter need to be encased. Even then much dust is caused at the issue into

the vats, though this may be partially lessened by a fine spray of water. It is admitted at Waihi that the escaping dust assays about twice as high as the average ore, which shows how very fine the gold must be, and what a large pecuniary loss this dust must represent. In some mills an attempt is made to minimise dust by covering the truck with an old filter-cloth, and dumping on to a spreader, so as to

FIG. 152.—CHARGING TANKS.

reduce the shock of impact and disturbance of contents. A "traveller" is also occasionally provided, to enable the truck to be emptied over any portion of the vat, and thus avoid raking.

From accumulations lying at a distance from the works, tramways and trucks, with steam or hand haulage, are most in favour, though cable and bucket tramways for long distances and conveyor belts for short ones, with excavators or dredgers for filling, would in many instances be found much

cheaper both in first cost and in operation, except when native labour is obtainable at very low figures and fuel is expensive.

Fig. 152 shows an inclined tramway with winding rope for hauling truck of tailings from the dump at the White

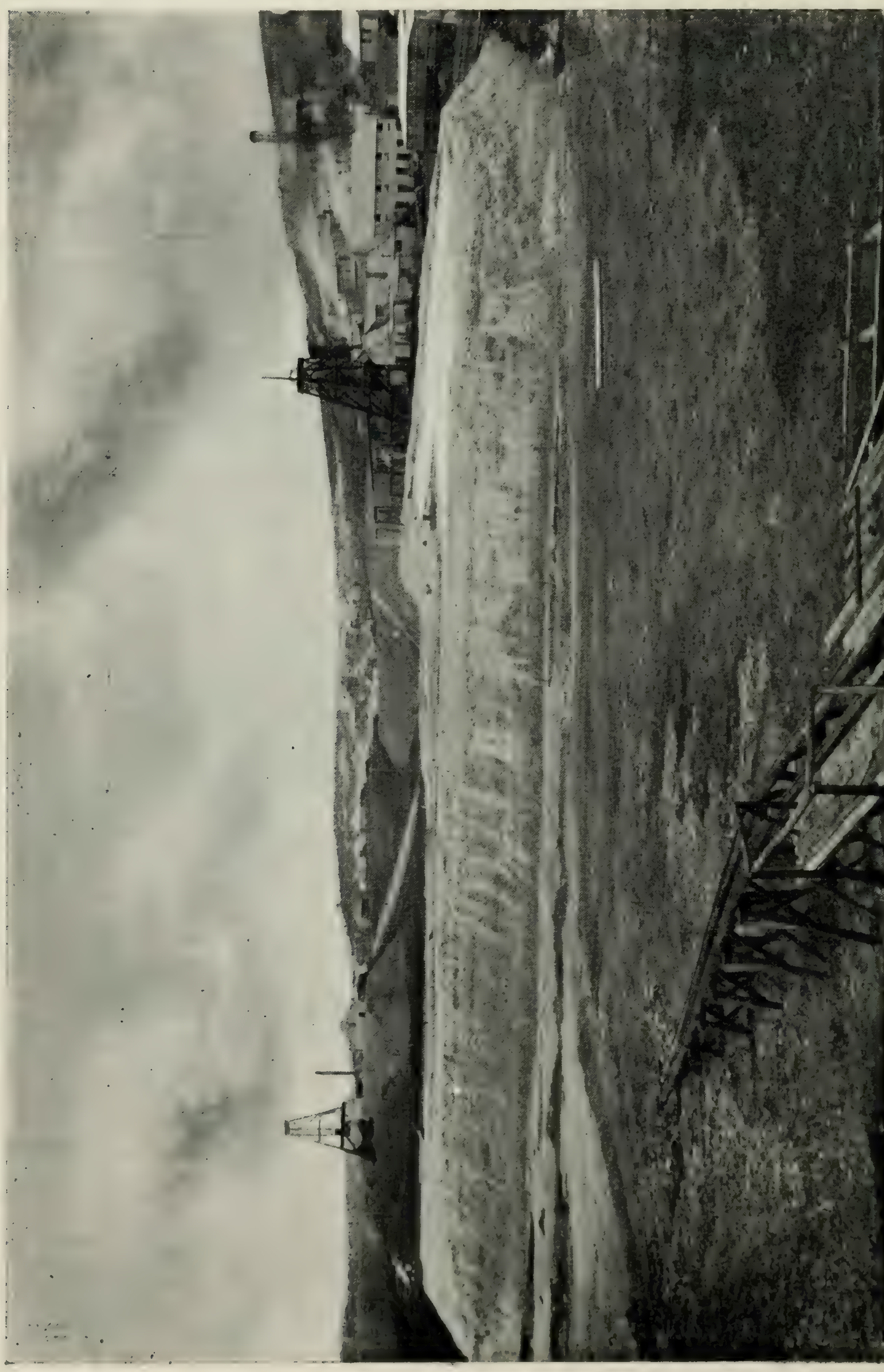


FIG. 153.—TAILINGS.

Feather Main Reef, W. Australia. By suspending a sufficiently heavy weight by rope and small sheaves from a little carriage to run to and fro on the flat section of the tramway, the emptied truck can be automatically started back on its downward course; as the truck arrives full at the top of the incline, it encounters the balance-carriage and pushes it ahead to its destination; as soon as tension is taken off the hoisting rope, the balance carriage exerts its pressure to start the empty truck back, the hoisting-engine serving as a brake to control the downward speed. Much labour can thus be saved.

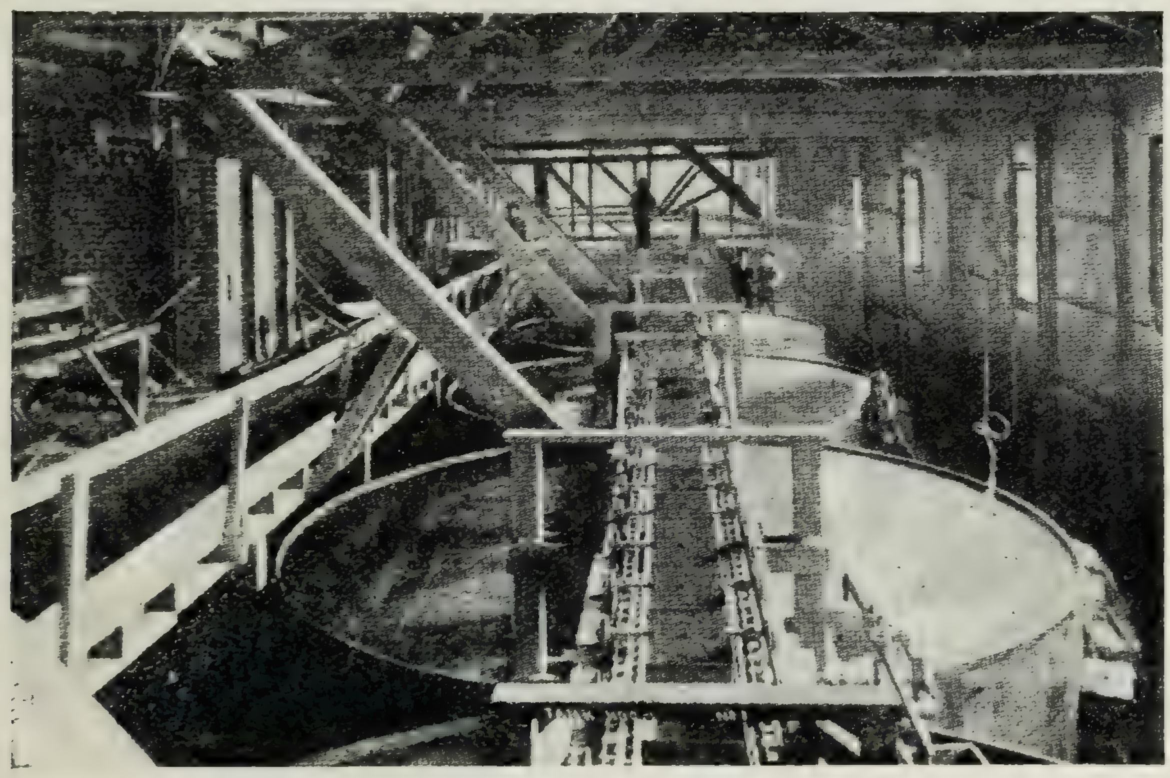


FIG. 154.—CONVEYING TAILINGS.

Such a contrivance is adopted at the Wentworth plant, Lucknow, N.S.W., and works well. In this case, a large deposit of old slimes (Fig. 153) has to be dealt with before the main heap behind can be reached. The foot of the tramway dips into slimes 5 ft. deep. These are hauled by trucks and a winding engine to the platform (in the foreground), and then fed into the trommel (on the right) which rotates in a tankful of water beneath. From the tank, in which they are converted from a very stiff and tough clay into a soupy consistence, they are raised by a centrifugal pump to the leaching vats shown in Fig. 150.

In Fig. 154 is shown a leaching plant equipped with a

Jeffrey scraper conveyor. The scrapers are provided with axles and truck wheels, the wheels running on suitable tracks outside of the trough, and being of sufficient diameter to support the scrapers in the trough without allowing them to touch the bottom, thus preventing friction, noise and wear, as well as reducing the power required. This arrangement also removes the chain and wearing parts from contact with the gritty material, greatly prolonging the life of the machinery. The trough is provided with openings or gates, by which the material may be delivered as required at various points. Revolving swivel spouts with telescopic extensions are often attached to the gates to secure a better distribution of the material. The lower or return chain runs beneath the vats, and is frequently used to carry away the refuse. These appliances are great labour savers.

At the new Arequa mill, Elkton, Colorado, a conveyor made by the Jeffrey Co., which is a modification of the trough system, has been installed. That conveyors can be made to do the work of filling old tailings or dry-crushed ore more cheaply and with less compression in the vat than is possible with hand shovelling and trucking seems to admit of no doubt whatever from an engineering point of view, and it is surprising that this method has not received greater application. Even if the cost were heavier than manual labour—which it could only be under exceptional circumstances—the even and gentle delivery of the conveyor is worth it, as the charge is left in a free condition which is in marked contrast to that brought about by dumping a truck-load of ½ t. or even 1 t. at a time from a considerable height.

The Eureka mill, on the Carson river, Nevada, is one of the most recent and advanced in America, and is built to work on low-grade tailings from the Comstock lode, which carry much of their value in silver, necessitating economic handling. Here conveyors are used with great success. The tailings are very varied in composition and much scattered. They are all conveyed to a bin of 150 t. capacity, and from 4 discharge hoppers the different products are automatically fed into a conveyor in the quantity required. In their passage

to the plant, they are mixed, by means of a branch conveyor, with the required quantity of coarse sand to form a uniform product suitable for percolation. As the sand passes along, it receives the requisite quantity of lime from an automatically discharging hopper. Finally, it is dropped into the boot of an elevator, is lifted 47 ft. into the ventilator of the tank house, and is discharged by means of shoots into the vats. The elevator is a slow-speed perfect-discharge chain-elevator, with steel-bushed chain and large buckets, and discharges into the conveyor on the upper floor from each bucket the exact quantity delivered at a stroke by the lower conveyor. The upper conveyor is located high enough above the vats to admit of placing 45° shoots under the conveyor-box, which deliver the material 3 ft. 6 in. above the vats. There are three shoots to each vat, which fill the vat (150 t.) in about 4 hours. The sand must be hand-distributed, but the conveyor-attendant does this work with little trouble. The conveyor is a reciprocating machine with pendant, hinged flights fixed on a horizontal tubular bar, the whole operating in a narrow wooden box. These flights are rigid on the forward motion, but fall back on the return.*

With labour at 10*s.* a day (10 hours' shift) in Colorado, it is computed that, if the distance be not too great, 2 men, with a truck holding 1 t. of pulp and a properly laid track, can load 10 t. per hour into a 20-ft. vat, including weighing and distribution, which would make the cost about 2½*d.* per ton. At Rio Tinto, Spain, according to H. F. Collins,† there is no difficulty in getting large gangs of men to shovel 30 t. a day per man, which is little more than half the Colorado estimate of human capacity, but Spanish wages are less than half the Colorado scale, though the working hours are probably longer, say, 12. On the Transvaal, it is estimated by Alfred James‡ that "a good man can shovel into trucks about 20 t. of tailings per day of 8 hours," which seems to accord pretty well with Rio Tinto results. According to M. Eissler,§ the

* Bosqui, p. 184.

† Trans. Inst. M. & M., iii. 413 (1895).

‡ Ibid., iii. 406. § Ibid., iii. 54.

cost of charging tailings and discharging residues at the Robinson mill is *7d.* per ton, and it "generally stands in the a/cs of other works at about 1*s.*" Reverting for a moment to the Colorado figures, they would imply that each man of the pair shovels 5 t. or 10,000 lb. per hour, which is equivalent to 166 lb. per minute, or a shovelful weighing 16½ lb. every 6 seconds. They would be remarkable men who could keep that up for 10 consecutive hours, without counting the truck-pushing, weighing and distribution which they are also supposed to do. Probably about half the stated duty is nearer the mark, making the cost 5*d.* per ton. There is thus ample room for economising by the adoption of mechanical arrangements.

In working up the old accumulated tailings at the Montana mill, near Marysville,* an effort is made to get material as nearly as possible uniform in fineness, and to avoid lumps of hardened slime. To accomplish this, the bed is ploughed, harrowed, and worked over continuously in order that drying and pulverisation may be ensured, and it is in contemplation to add pulverising machinery. Loading the dry pulp for conveyance over 2½ miles to the cyanide plant, is effected by a machine known as the Fresno buck scraper, which deals with 400 cub. yd. daily, with an average haul of not less than 100 ft., employing 6 men and 12 horses. The use of the steam shovel in this work is practically prohibited by the fact that the material should be somewhat dried and pulverised on the surface of the bed before loading, in order to secure the best working results. The cars used are of 3 t. capacity, with bottom discharge, and 16 loaded cars make the train for a 22-ton locomotive. The tailings, after leaving the cars, pass in an almost continual stream to the sheet-iron lined bins, and out of the gates to a 24-in. four-ply belt conveyor, which conducts them to a revolving chute or distributor; and this in turn fills a vat 38 ft. diam. by 9 ft. deep with 400 t. of tailings in about 8 hours. The great advantage of filling a tank in this way is that it gives a charge of uniform permeability. There are four of these vats, each with its bin,

* C. W. Merrill, En. & Min. Jl., May 16, 1898.

conveyor and distributor, and one is charged daily, thus giving 4 days to complete the treatment of each charge.

At the new mill of the Great Northern Co., at Gilt Edge, Montana, a 2-ton car is used for charging the tanks (28 ft. diam. and 3 ft. deep), being run on an endless rope; 1 man thus fills a 70-ton tank in less than an hour.

The cable tramway used at the Meyer & Charlton mill in filling tanks is shown in Fig. 155.

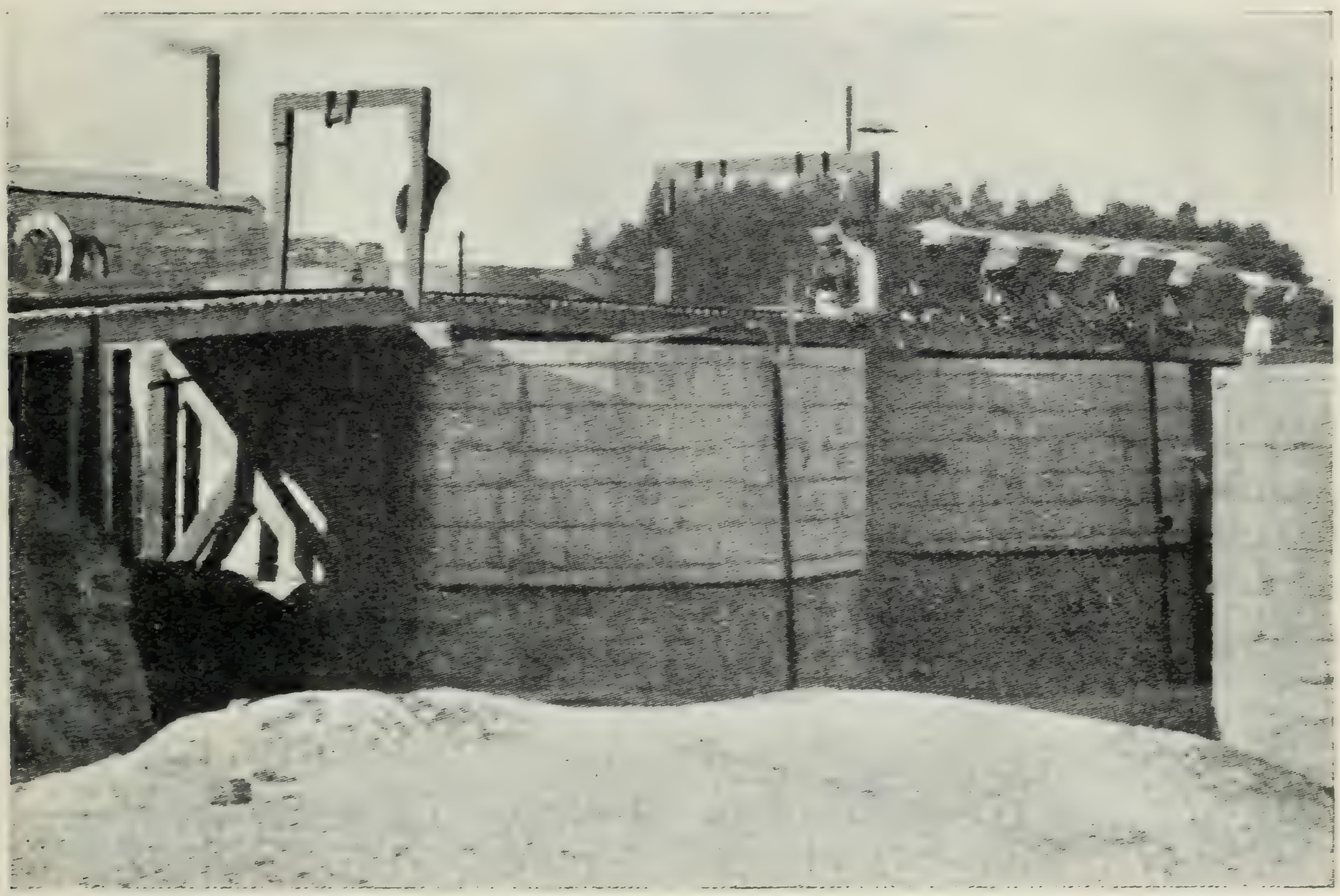


FIG. 155.—CHARGING TANKS.

The Princess mill also adopts truck filling, Fig. 156.

When the settling tanks at the Simmer & Jack are full, they are discharged through bottom doors into cars, which are carried by inclined cable railway to the leaching tanks.

In using centrifugal pumps, certain precautions are necessary. The pump must be firmly bolted down to a wooden or brick foundation, perfectly level with the pump spindle, and in a line with the driving pulley. The driving belt should be of good length and width, as a tight one will

throw extra strain on the spindle, and entail loss of power in friction, besides wearing out much sooner. In most cases a foot-valve is necessary to retain the water, as these pumps require charging before starting. This, when once done, seldom demands repetition. Belts should be as near hori-

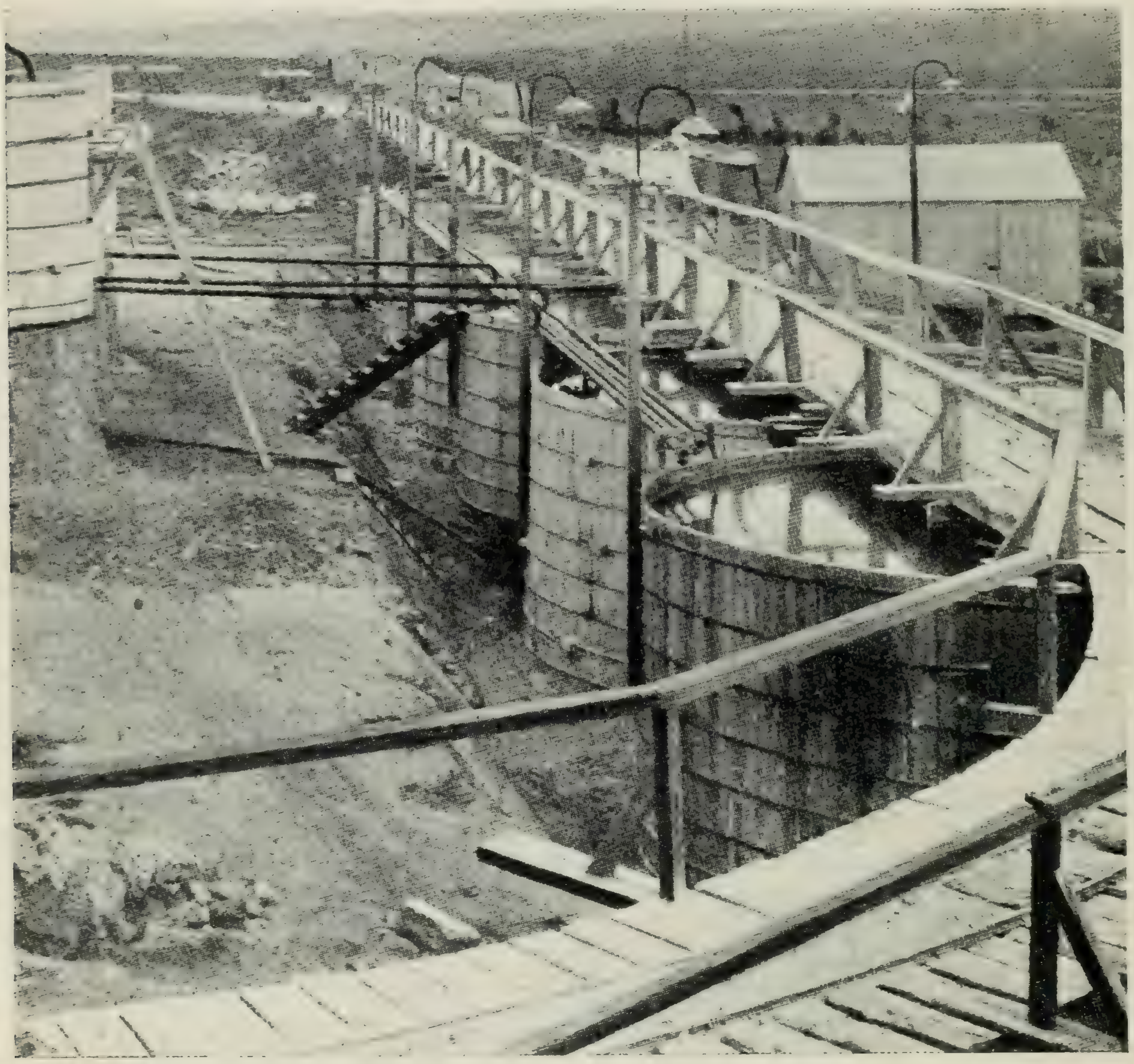


FIG. 156.—CHARGING TANKS.

zontal as possible, and not exceeding 45°. For *very* large sizes, a flap-valve on the delivery is better than a foot-valve, and the pump is charged by exhausting the air. The pump being fixed in the most convenient position, the suction pipes and foot-valve should be connected, with rubber rings between the flanges, it being of the utmost importance to have all

joints air-tight, as a small leak will impair the efficiency of the pump. The whole of the foot-valve and grating must be well immersed in the pulp, or water will be pumped and no solids. The pump and pipes must be charged with water, and allowed to remain some time to detect any leakage. If satisfactory, the pipes can be connected, care being taken to avoid bends, as owing to the great velocity of the stream they entail much loss in friction and extravagant wear of pipes. When suction pipes run any great distance horizontally, they should be placed with a gentle rise towards the pump, so as

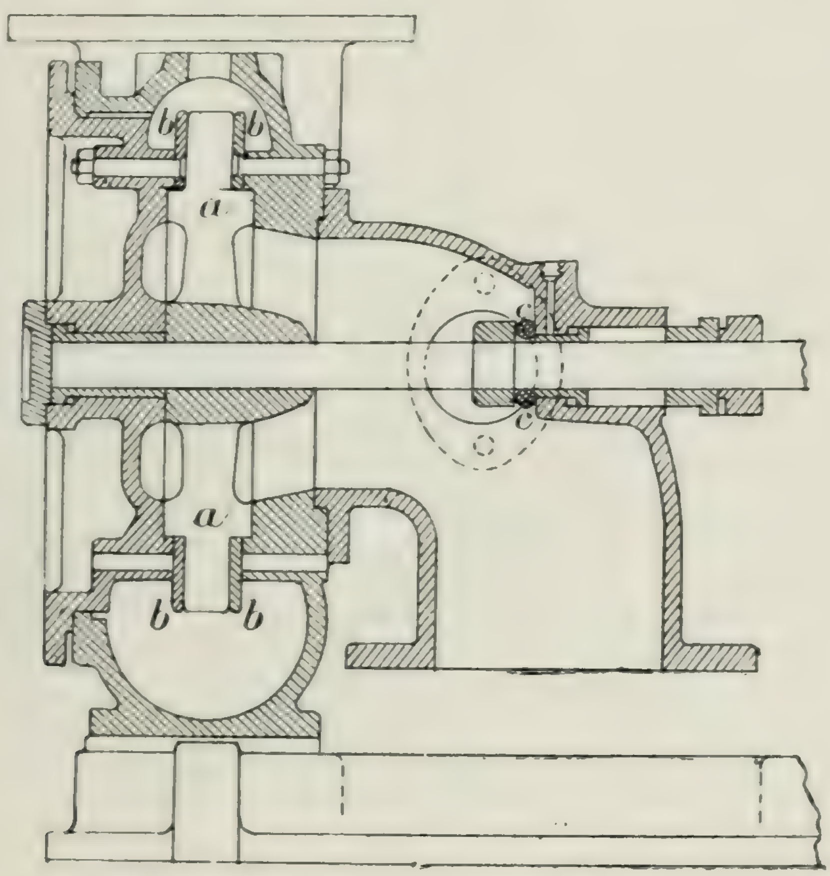



FIG. 157.—CENTRIFUGAL PUMP.

to prevent any accumulation of air. If this is not possible, an air cock is inserted in the highest point to allow it to escape. Before starting, the glands should be attended to, precaution being taken that they are not screwed too tight. Power should be applied gradually. With proper care, centrifugal pumps do excellent work at very low cost. They are most successful on lifts up to 20 ft., but may be worked up to 40 or 50 ft., with an efficiency of about 60 %.

The best pump of the kind is that made by Tangyes, as shown in Fig. 157. In this, the revolving arms *a* work

between renewable side-plates *b* of the most durable steel, and provision is made for gaining ready access to the interior for effecting renewals without interference with any pipe connection. A rubber ring also is introduced at *c* to exclude grit.

When a valve is used for admitting air among the pulp for aëration purposes, it should be placed in the delivery pipe, and connected with a compressor. To admit air at the suction is to greatly reduce the efficiency of the pump and increase the cost of pumping.

Plunger pumps work very well when provision is made against grit by introducing a small stream of water under pressure below the gland of the plunger in the plunger case; otherwise the packing of the plunger is rapidly destroyed, and the plunger is cut and scoured away by the sand.

A "spiral" sand pump, by Frenier & Le Blanc, at the Congress mill, Arizona, raises 90 t. of wet-stamp pulp 15 ft. every 24 hours. It consists of a spiral ribbon of $\frac{3}{16}$-in. steel plate, in form like a dismounted clock-spring, on each side of which is a disc of steel plate, which is riveted to the spiral, making a continuous air-tight joint. There results from this a spiral tube of steel, which has a rectangular cross section of equal area throughout. The outside diameter is 4 ft. It is mounted on edge with horizontal shaft, makes 20 rev. per minute, and receives water and sand as the open end descends into the well or sump. The sand and water work their way inward by the spiral path till they reach the hollow shaft, which connects with a stand pipe, carrying the sand and water up a maximum of 20 ft. In the construction of the pump, there would seem to be considerable difficulty in making a really tight joint between the spiral and the sides. Further, the economy of power is more than doubtful, as the friction is excessive. Experiments were made at Lucknow on a similar contrivance but of simpler structure, and the amount of motive power required was such as to condemn the arrangement entirely.

Quite a common device is a tailings wheel of sufficient diameter, either having a number of wooden compartments in

its periphery (for large wheels) or carrying a large number of steel buckets on its circumference (for small ones). As this slowly rotates, the compartments or buckets pick up the pulp from a well or pit and discharge it into launders on either side. Wheels have been built 60 ft. diam. Their maximum efficiency is about 75 % of their diameter, but it is safer to count on about 66 %. Steel buckets wear out very rapidly. Examples are shown in the annexed illustrations, Fig. 158 representing a single wheel at the New Primrose, and Fig. 159

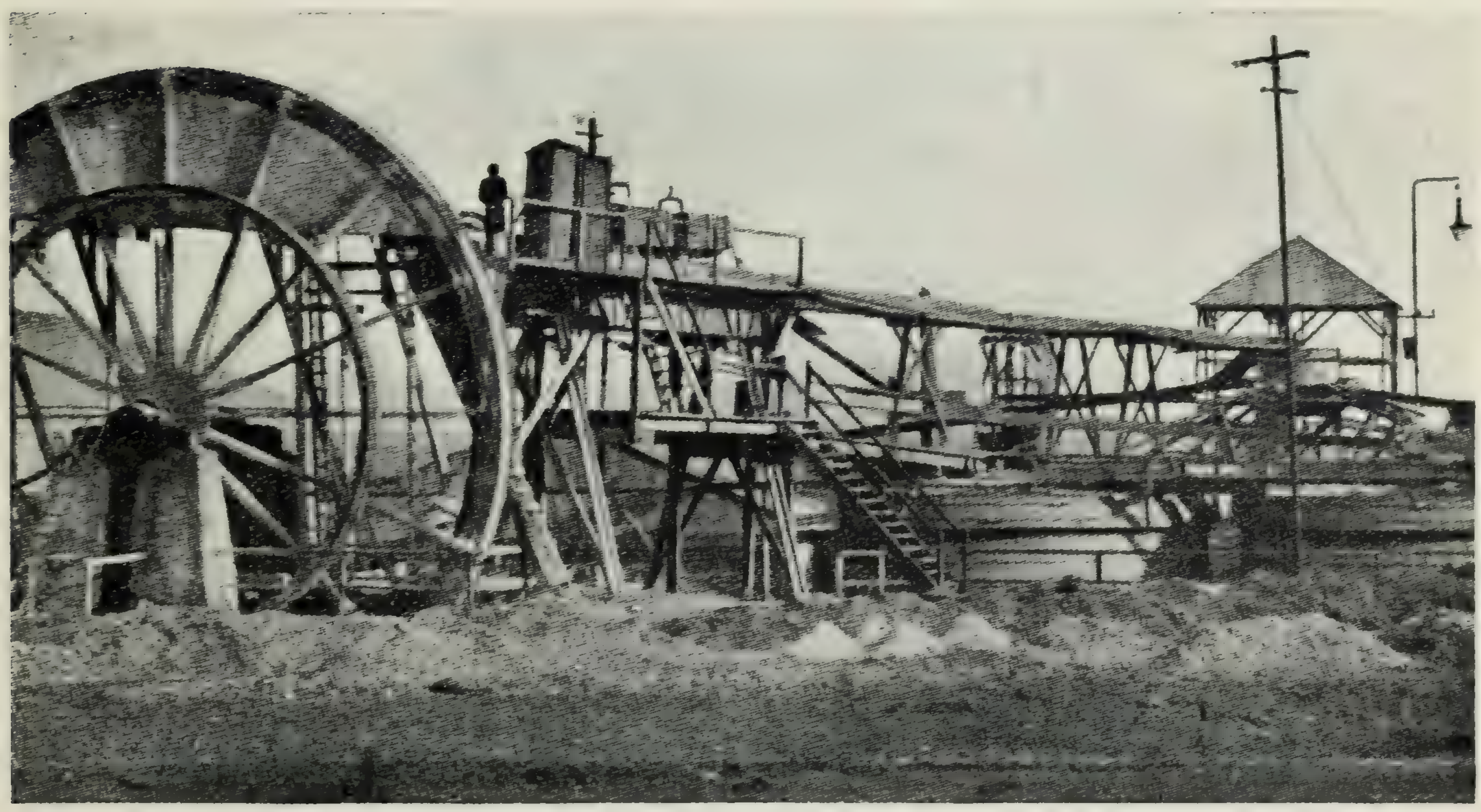


FIG. 158.—TAILINGS WHEEL.

a double one at the Crown Deep. No little skill is required in building them to avoid warping, in which case all sorts of difficulties arise. The central hub is cast around the inner ends of the radiating arms, but in the author's experience, even with a 14-ft. wheel, this is by no means an easy job.

*Filling.*—All tailings consist of at least two distinct classes of material, one being coarse and easily permeable by fluids (known as "sands"), while the other is fine and impermeable or claylike (termed "slimes"). In old accumulations, these will have become naturally separated to a very large extent, the sands settling immediately they escape from

the launder, the slimes travelling far before they are deposited, and midway will be a mixture of fine sands and coarse slimes. Each deposit can be drawn upon as required with more or less distinction from the others. But current tailings as delivered daily by the battery contain both sands and slimes, and, unless some form of concentrator has been used,

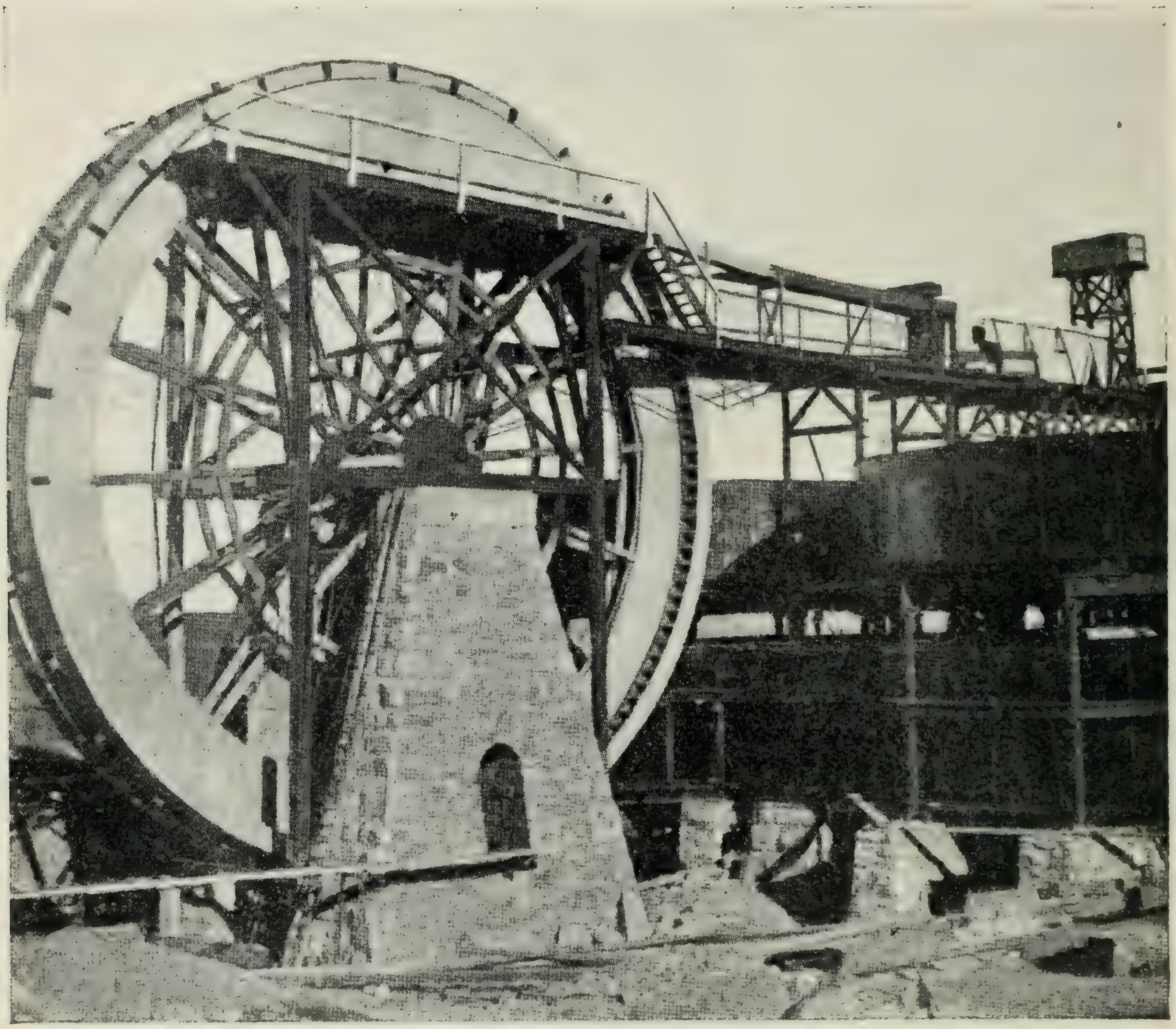


FIG. 159.—TAILINGS WHEEL.

a third product consisting of pyritic material, some of which will be among the sands and some with the slimes, the relative proportions varying at every mine.

It has come to be recognised that the marked physical and chemical characteristics of these three classes of product require that their treatment shall be conducted on separate lines. The pyritic matter is usually much richer in gold

than the remainder, and requires longer exposure to cyanide solution in order that all the gold in it may be attacked. The sands admit of simple lixiviation with weak liquors, being in a condition which renders their mass porous; while the slimes, which in some cases are by far more valuable than the sands, and in others are so poor as not to repay treatment, cannot be leached at all without agitation.

Preliminary investigation of the most careful kind is necessary to determine for each ore the proportions, features, and economic values of each class of material, so as to decide how far their separation is to be carried. The pyritic matter may be so valuable that the best possible concentrating machinery may be justified, and the concentrates be so rich that chlorination or smelting must be resorted to for them; or it may consist of coarse-grained barren pyrrhotite or marcasite not worth isolating from the sands; or again, of copper pyrites or silver sulphide which it may be desirable to remove so as to minimise the waste of KCy liquor which its inclusion would involve. Then it is necessary to know whether it will pay better, in the separation of sands and slimes, to allow some of the latter to contaminate the former, offering greater impediment to rapid leaching, but reducing the bulk of slimes and the scale of the plant necessary for their treatment; or whether the converse is desirable, viz., to free the sands absolutely from slimes, and perhaps discard the one or the other, or prepare for comprehensive treatment of both.

On the results of these investigations are founded the various methods of "filling" the vats, using that term to embrace the accessory operations of separation or classification of the product. As a matter of fact, all cyanide experts are agreed that some classification of wet-crushed tailings is advisable, and there is probably no important works in existence where it is not provided for. The main point at issue is whether the operation shall be performed in the leaching tanks or in specially contrived apparatus on the lines of the hydraulic classifiers described on pp. 324–337. For the first-named method, the term "intermediate filling"

has been coined in S. Africa, being suggested by the fact that an extra vat is placed in an intermediate position between the tailings launder and the leaching vat. In contradistinction, the second method is called "direct filling," there being only the usual leaching vat. A controversy has long raged in the Transvaal on the respective merits of the two systems, the facts being that the conditions of each mine must determine which method of classification is best adapted to its particular needs. In both cases they are simply forms of—

*Classification.*—The first attempts at intermediate filling were made in the Transvaal * by running the battery tailings to the centre of a circular vat, and allowing the overflow to take place at one point. This did not prove successful, because the sand piled up in a central conical heap, and the slimes settled in the deeper water around the sides of the tank. The next plan was to run the pulp into the vat through a series of stationary launders, delivering at several fixed points. This improved the distribution, but the result was still unsatisfactory. Then, in order to give uniform overflow at every point of the periphery of the vat, a circular trough was fixed round the top to collect the overflow, and deliver it to a launder. Each of these alterations was a step in the right direction, but the system of settling could not be considered successful until after the introduction of an automatic revolving distributor, on the principle of the garden sprinkler.

This consists (Fig. 160) of a central casting *a*, with a vertical spindle *b*, revolving in a footstep bearing; the casting carries a conical hopper *c*, and a number of radial pipes *d* with bent ends. The distributor is fixed on an iron column *e* in the centre of the vat. The bends at the ends of the pipes cause the apparatus to revolve by the reaction of the pulp as it leaves them. Each pipe has a different length, in order to distribute over a number of concentric circles. It was found that the slimes collected in narrow rings between the outlets of the pipes, giving rings of clean sand alternately with rings of slime; this difficulty was overcome by attaching flattened nozzles to the ends of the pipes, causing the pulp to

* M. Eissler, Trans. Inst. M. & M., iii. 50 (1895).

spread over a wider area, and also by increasing the number of pipes. The hopper is covered with a coarse screen, so as to prevent chips or leaves entering and choking the pipes. The diameter of the discharge pipes is $1\frac{1}{2}$ to $2\frac{1}{2}$ in.

In intermediate filling, it is necessary to fill the vat with clean water before admitting the pulp. If this is not done,

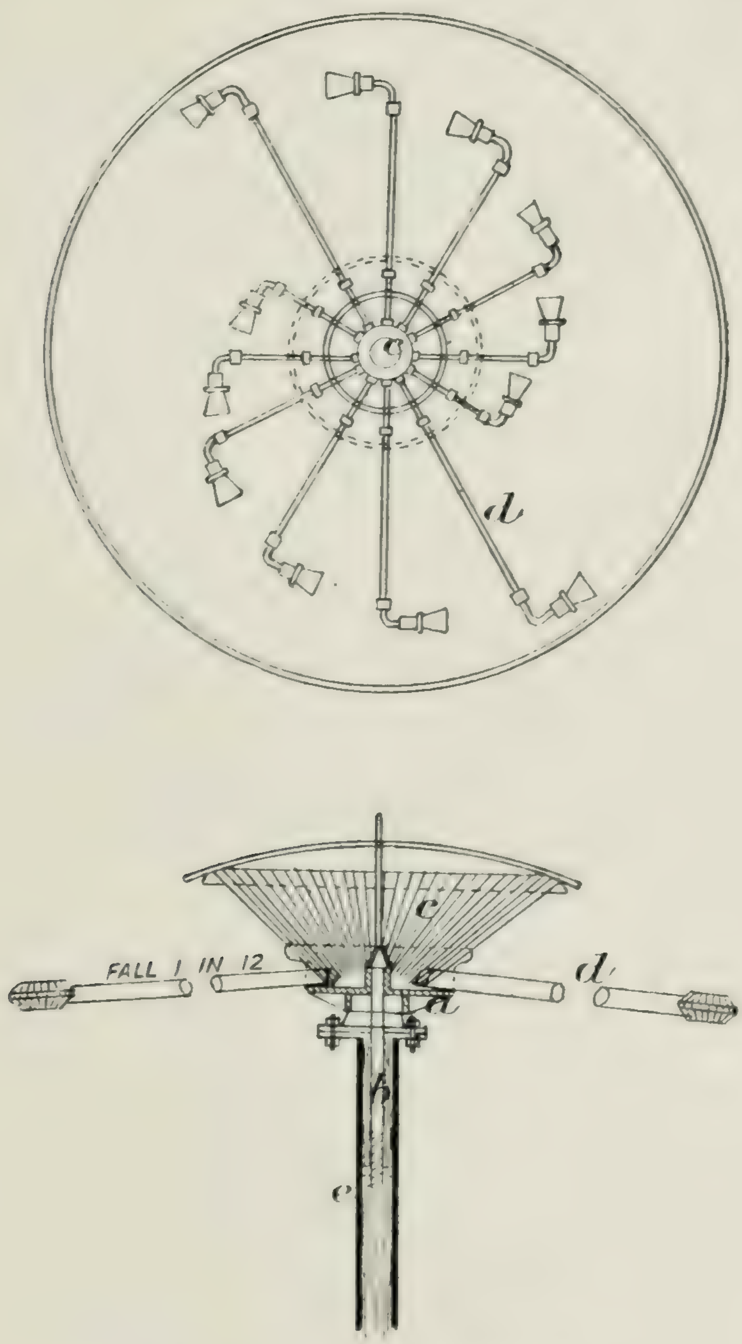



FIG. 160.—DISTRIBUTOR.

the water is practically stationary, and a constant settlement of slimes takes place, until the vat is full and the overflow begins, in which case the tailings in the lower part of the vat will always be more slimy than those in the upper part. For the same reason, it is essential that the overflow be continuous until the vat is full of sand, for, if any stoppage takes place,

slime settlement in excess occurs, and a complete layer of slime is formed across the vat, which prevents the overlying sands from draining dry. Therefore, when the battery is stopped, a quantity of water should be supplied to the vat equal to the full volume of pulp ordinarily flowing. When the pulp is admitted into a tank previously filled with water, the light slime remains in suspension, and overflows into the annular ring which surrounds the tank at the top ; from this the slimes are carried by a launder to storage pits or agitation tanks (for treatment), or are allowed to run away as waste.

When the vat is filled with tailings, the outlet pipe below the filter is opened, and the water is allowed to drain off, which takes about 15 to 24 hours. When holes are dug down to the bottom discharge doors, water again commences to flow from the outlet, consequently it is advantageous to dig these holes about 6 hours before discharging.

A very important matter is the proper size of intermediate vat to be used for a given tonnage crushed in the battery. It may be desirable to catch as large a quantity of slimes with the sands in the tailings as is possible, without rendering the product unleachable. Then if the vats are too small, they will send away too much fine sand with the slime ; while, if too large, they will catch slime in excess. The great difficulty with intermediate vats is to get the last 1 to 2 ft. near the bottom properly drained. If the tailings are discharged and transferred to the leaching tanks in a very wet condition, the excess of moisture dilutes the cyanide solution.

At some S. African mills, the intermediate vats separate 75 to 80 % of good leachable sand containing 12 % moisture after draining 18 to 24 hours. The following sizes of intermediate vats are erected :

(*a*) for treating 120 t. per day, 4 vats, each 20 ft. diam. × 8 ft. staves.

(*b*) 70 t., 2 vats, 20 ft. diam. × 14 ft. staves.

(*c*) 70 t., 2 vats, 20 ft. diam. × 8 ft. staves.

(*d*) 85 t., 2 vats, 20 ft. diam. × 7 ft. staves.

(*e*) 330 t., 6 vats, 24 ft. diam. × 11 ft. staves.

When all the pulp is run into one vat, only about 66 % is

caught, but the whole of this is clean sand and drains readily. If the pulp is run into two vats, about 80 % is obtained.

From the intermediate tank, after the water has been drained out, the ore is discharged through bottom doors into trucks, and taken to the leaching tanks, which are sometimes on a higher and sometimes on a lower level, involving power haulage in the one case and simple gravitation in the other.

It is claimed that, by means of distributors, 75 to 80 % of coarse and fine sands, with some slimes, are collected in the intermediate tanks, the bulk of the slimes escaping with the effluent water, and being practically free from sands ; and that, when the intermediate vat is discharged, the sands get thoroughly mixed up and aërated, thus being put in the best condition for treatment by cyanide. The expense of transferring the tailings from the intermediate tank to the leaching tank is slight, and is considered to be warranted by the free leaching condition ensured in the charge.

"Direct filling" consists in passing the pulp leaving the plates into a hydraulic separator. The pulp is divided into two streams,—one overflowing, carrying slimes with very fine sands ; the other being mainly coarse sands, with some fine sands and slimes, which are conveyed by means of a rubber hose to the leaching tanks, in which labourers effect the even distribution of the pulp, by moving the hose about to different parts of the vat. The water passes off by adjustable gates fitted inside the vats, carrying with it fine sands, slimes, and some coarse sands. The advantages claimed for this process are that it treats pyritic tailings with the minimum of oxidation, as they are not exposed to the action of the air from the time they leave the battery ; and second handling of the tailings before treatment is avoided. On the other hand, the tailings pack tightly in the vat, and consequently do not drain completely, so that a diffusion of the first cyanide solution takes place at the commencement of leaching ; and the distribution of the sands and slimes is said not to be so even, so that some sands escape treatment, being protected by impervious layers of slime, the cyanide naturally escaping by the paths

of least resistance. In leaching tanks where uneven distribution of slimes and sands takes place, the slimy portion will not drain off; on discharging such a tank, it is easily noticed that the streaks of slime are saturated with moisture, and are still gold-bearing, whereas from the sandy portion the solution has drained off. The importance of even distribution and mixture of the pulp is very great.

*Concentration.*—The separation of pyritic material from tailings previous to cyaniding is in almost all cases a course to be commended, the degree of concentration depending on the proportion and characteristics of the sulphurets. This seems more rational than concentration after cyaniding, though that is sometimes done. Pyrites are more likely than any other portion of the ore or tailings to contain ingredients which, being insoluble in water, cannot be removed by preliminary washing, yet are acted upon by and cause a loss of KCy solution, which is in itself sufficient reason for their elimination. At a few of the Transvaal mills, vanners (p. 337) have been used for separating such matter as would concentrate in this way, but in general it has been found sufficient and less costly to pass the pulp through hydraulic classifiers, such as pointed boxes and troughs. As an example of this practice, it may be mentioned that at the Henry Nourse works, the largest and most modern in S. Africa, it is calculated that the pointed troughs will isolate 25 % of the tailings as concentrates, with a value of 15 dwt. per ton, and these will receive special treatment. As contrasted with the vanner, the hydraulic apparatus saves a much more bulky concentrate of considerably lower grade. Thus while the vanner product may assay several oz. per ton, and quite justify the cost of chlorination or smelting for recovery of its contained gold, the box-collected material will be diluted as it were by much coarse and comparatively valueless sand, bringing down the assay value of the whole perhaps to a figure that debars the application of a costly secondary treatment, while it is not by any means too well adapted to cyaniding. But experimental treatment can alone decide this point for each ore.

*Sampling.*—With dry crushed material, sampling is usually effected by drawing a small portion of the contents from each truck on its way. The tool used for this purpose is a length of $\frac{1}{2}$- to 1-in. gas pipe, filed to a cutting edge at one end, and provided with a $\frac{1}{4}$-in. slot for its whole length, so that the charge can be scraped out if it should pack so tight as to resist shaking out. The samples withdrawn are deposited in a locked box, and accumulate until the vat is filled (or emptied, as the case may be), when the assayer or his assistant takes the parcel and quarters it down in the usual manner, removing sufficient for at least three assay samples, and returning the remainder. When the residues are to be sluiced out of the vat after treatment, the sample is taken by probing with an iron of the requisite length, care being taken to avoid having an undue proportion of the upper layers, which always exhibit most complete treatment. Generally a special sample is taken of the last 6 to 18 in., wherein results are below the average. With charges or residues such as slimes, which have not sufficient cohesion to be held in a sampling iron of this description, a most effective and simple contrivance is a piece of gas-pipe, without a slot, but provided with a screw cap which makes it airtight. The probe is inserted with cap off, and fills with liquid pulp ; then the cap is adjusted, and the probe is withdrawn with its sample. It occupies a little more time to tighten and loosen the cap at each probe, but is the most accurate method of sampling a tankful of slimes. Automatic sampling mechanism for adjustment to tailings launders has been described on p. 179.

*Discharging.*—In very many mills, the discharging of vats is accomplished in the same way as the charging, viz. by hand labour and trucks, the trucks being variously arranged to enter the vats or to run beneath or alongside them. Bottom or side discharge doors (see p. 515) are used accordingly.

Shovelling out on to a conveyor belt is practised at some mills. The practical economic limit for manual labour is 4 ft. in height and 8 ft. in distance.

Sluicing out is, however, much to be preferred for several reasons, when water is available for the purpose, even though

it be necessary to pump that water back again for re-use and to gain sufficient "head" or force. At the Drumlummon works, Montana, the tailings are sluiced out by 2½-in. hose with water at 60 ft. pressure, through 4 side-issue doors and one in the centre of the bottom, a 400-t. tank being emptied in less than 3 hours, and the operation costing under 1*d.* per ton. No manual labour can compare with this. Even with a 1-in. jet, under a head of 50 ft., one man can sluice out 25 to 50 t. per hour. Where gravitation pressure is not available, a force-pump is equally effective—though, of course, more costly.

In addition, by sluicing out, provision can be made for the arrest of any coarse gold or amalgam or untreated pyrites, which may have evaded capture by previous operations, by passing the stream over amalgamated plates, and placing riffles, blanketing, etc., in the launder. Such safeguards will always more than pay for themselves.

Relative local costs and supplies of labour, fuel, and water must determine the choice of method.

**Storage and Circulation of Liquors.**—In order to carry on continuous work there are required stock tanks or reservoirs for solutions of various degrees of strength ; a dissolving tank for the KCy ; store tanks for such alkaline washes as may be used ; pumps for raising liquors, washes, and water ; and a system of pipes and cocks for carrying KCy solutions, washes, and gold liquors.

*Stock tanks* should be of sufficient capacity to contain at one time all the solutions—strong and weak—in circulation in the works. The dimensions thus depend on length of treatment, and proportion of solution and washings to ore treated ; but, as a general rule, accommodation is provided equal to one-third of the cubic contents of the leaching tanks. Thus, for 6 leaching tanks there would be 2 stock tanks of the same dimensions, the one for strong and the other for weak solutions ; but where the weak solutions greatly exceed the strong in volume, the comparative sizes of the respective reservoirs must be altered. Stock tanks are constructed in precisely the same manner as leaching tanks, except that

they have no doors and no filter bottoms. When placed at a less elevation than the leaching tanks, so that solutions may flow into them direct from precipitation, they are termed "sumps," and it is usual, in this case, to have at an elevation above the leaching tanks, an additional smaller tank of such capacity as to contain the amount of solution required for any one charge. Thus if a plant treats 100 t. a day, and uses an amount of strong solution equal to $\frac{1}{3}$ the weight of the ore, the smaller tank would have a capacity of about 35 t. of solution; or if the plant treated 200 t. a day, charging two 100-t. vats, one in the morning and one in the evening, the small tank would still be of 35 t. capacity.

The advantage of having this upper tank for standardising is that the limited amount of solution required may there be made up to the desired strength, instead of having to keep all the solution in the sump up to the normal strength; and also that one does not require to be running the pumps so frequently, as the solution, once pumped up to it, may be allowed to run on to the leaching tanks at such a rate and in such quantities as desired. This in practice is of the greatest advantage, and minimises the inconvenience consequent on a breakdown of the pumps. It is fitted with a tell-tale, consisting of a float actuating an ordinary variable-level gauge or indicator, which, working against a scale of tons, shows the contents at a glance.

Another plan is to have the two stock tanks at a higher level than the leaching tanks, when the solutions flow from the latter into a small sump, whence they are continuously pumped up into the stock tanks. This is not such good practice, as it necessitates the constant employment of power for pumping, and the total suspension of operations in case of a breakdown to the pumps, unless a duplicate set is provided. It also requires the whole of the solution in the upper "strong" tank to be kept up to its normal strength, and this is inconvenient when it may be required to treat exceptional charges with different strengths of solution. It has the advantage of lessening the elevation necessary for the leaching tanks, or of alternative excavation for the sumps.

Where alkaline washes are required, an additional stock tank must be provided for the storage of these.

If vacuum suction is used (p. 520), two small tanks will be needed, each capable of holding the contents of the receiver, and connected with the strong and weak precipitators respectively. This permits of continuous working. These tanks are usually made square (as occupying less space), and shallow (so as to keep the receiver as low as possible, to prevent loss of suction power or increased elevation of leaching tanks).

Finally there must be a small tank in which the solid cyanide is dissolved. This is effected by placing the cyanide in lumps on a movable suspended tray kept near to, but below, the top of the solution contained in the tank; by raising or lowering the tray (by hand with a small lever and handle attached to the tray or basket, or mechanically by a little cam, pulleys and belting), the contents of the tank are agitated and the rate of solution is increased. The point of issue for the liquor must be a few inches above the bottom, to allow impurities such as carbide of iron to settle.

Stock liquor tanks are commonly made 20 ft. diam., and of varying depth, while both steel and wood are used in their construction, rarely masonry. A few examples are :—

| | Diam. ft. | Depth. ft. | |
|---|---|---|---|
| Robinson, Transvaal . . | 30 | 7½ | |
| Drumlummon, Montana . | 38 | 9 | California redwood. |
| do. . | 22 | 14¾ | do. |
| Golden Gate, Utah . . | 20 | 12 | steel on masonry. |

Each foot in height in a tank 20 ft. diam. represents 10 t. (2000 lb. each) of solution in round figures. The actual calculations are :—To obtain cubic contents of a circular tank, multiply the square of the radius by 3·1416 and the product by the depth of the tank: thus, for a tank 20 ft. diam. × 6 ft. deep,

$$10^2 \times 3 \cdot 1416 \times 6 = 1884 \cdot 96 \text{ cub. ft.}$$

and 1 cub. ft. water weighs 62·3 lb.

Then $1885 \times 62 \cdot 3 = 117{,}435 \cdot 5$ lb. ($\div 2000 = 58 \cdot 7177$ t.).

To prepare a stock liquor at ·3 % KCy, $(117{,}435 \cdot 5 \times \cdot 3) \div 100 = 352 \cdot 306$ lb. KCy will have to be dissolved in it,

adding any surplus which may be necessary to offset impurity in the commercial article used.

If, after treatment, this solution shows on analysis only ·16 % KCy, or

$(117{,}435 \cdot 5 \times \cdot 16) \div 100 = 187 \cdot 896$ lb. left in solution,

then the balance (viz. 164·410 lb.) must be made good by addition of fresh KCy.

*Settling tank.*—This most useful adjunct, which has been introduced by the Cyanide Plant Supply Co., is shown in Fig. 161. Hindrances to effective precipitation of bullion from solutions have been caused by the variation of the flow,

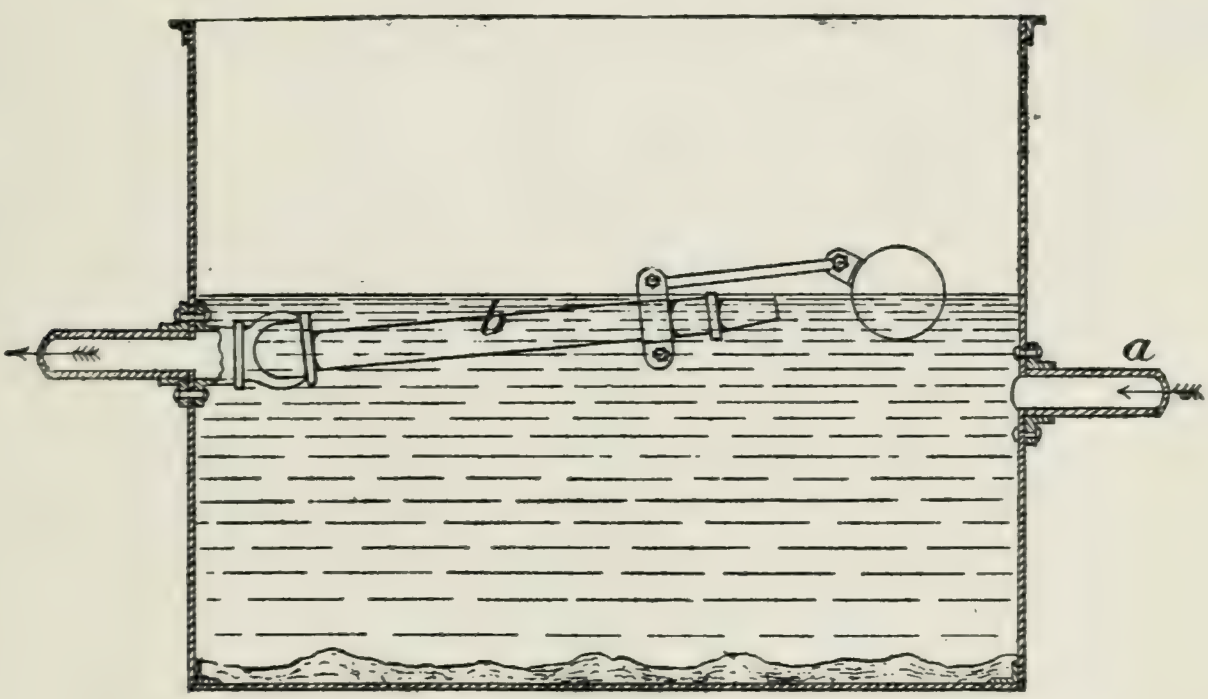



FIG. 161.—SETTLING TANK.

which is very considerable when the solution is turned on from two or more fresh charges at once. This necessitates a rapid flow through the zinc precipitation boxes (extractors), which are charged beyond their normal capacity, and the time of treatment, which is all-important, is thereby lessened, with the result that the sump solutions are rich in gold instead of being practically gold-free. The above apparatus prevents this.

It consists of a vat of suitable size, fitted with an automatic decanting apparatus, and placed in the circuit of the solutions before they enter the zinc boxes. Any abnormal inrush of solution at *a* thus only raises the level in the vat,

the lip of the decanting pipe *b* being always maintained by the action of the float at the level of the surface, and the solution is thus decanted into the zinc boxes at the normal rate. The apparatus further serves as an effective settler for any sand or slime which may escape the filters in the leaching vats, and for any precipitate which may be formed by reactions going on in the liquors. Such cause trouble by contaminating the gold mud, adding so much to the difficulties of the subsequent clean-up that the first compartment of the zinc box or precipitator is often used as a settling chamber. All this is avoided by having a settling tank in front of each precipitator, and clearing out the sludge from time to time.

Where a fitting-shop is on the premises, the decanting siphons may be home-made, but the elbow joint requires very accurate workmanship. Short lengths of 2-in. gas-pipe, with hardwood floats, were used at Lucknow by the author with complete success, and cost only a fraction of the purchased article.

With constant use, however, any joint will sooner or later give trouble owing to very fine pulp finding its way in ; and perhaps a better implement after all is good rubber hose, especially if it be strengthened by a wire coil, and then lined with rubber to give it a smooth internal surface.

*Pipe lines.*—There will need to be 4 systems of these :—

(*a*) A 3-in. service of liquor pipes from the upper stock tank to the agitators and leaching vats.

(*b*) A double service from leaching vats to precipitators, and to a vacuum receiver. The main lines are 3-in. piping with one 2-in. branch to each leaching vat, connected to both mains by means of a T-piece, so that the solution may be diverted into the "strong" or "weak" main and precipitator at will. These mains are now being replaced by V-shaped launders of wood dressed with paraffin, of which there must be 3, side by side. The 2-in. pipes from the leaching tanks are provided with hose and clamp, to enable the solutions to be led into the strong, weak, or waste precipitator. A great advantage of this is that one can see exactly what solution is

coming from each tank, and can sample it at any moment without interrupting the work. Of course, in this case, the vacuum will require a main for connection to the branches as before. This line of piping is usually connected with the first or solution system, so that, by turning a cock, the solutions may be conducted into the leaching vats from below the filter instead of from above: this is useful in some cases.

(*c*) A service connected with the pumps, and running from the sumps to the upper stock tank ; it is also connected to the two other systems, so that all three may communicate at will. Frequently the fresh-water and solution services have their main pipe line in common, but this cannot well be adopted where the residues are sluiced.

(*d*) A water supply must be led to the KCy dissolving tank, to the upper stock tanks, to each leaching tank and agitation tank (if any), to the precipitators for cleaning-up purposes, and to the assay and melting rooms. The main service to leaching vats and stock tanks should be 3 in. at least, except in very small plants, while 1 in. will mostly be sufficient for the remainder.

No less an authority than C. Butters prefers to have only one system of pipes for all purposes. This is of large size, say 4 in. Every tank is connected with a central centrifugal pump. Doubtless some saving may thus be effected in initial cost of plant, but it would need to be an enormous economy which would justify the inconveniences, delays, and risks of mistakes arising from a single pipe service.

Pipes, valves, and cocks should all be of iron, and made as free from rust as possible before putting in. Gas pipe is quite as good as steam tubing unless great pressures are contemplated, and is materially cheaper. Confusion of pipes, and liability to open or close wrong cocks may be avoided by simply painting each service a different colour.

*Pumps.*—The kind most favoured when driving power is conveniently available is the centrifugal (6 in. size generally). For direct driving by steam or compressed air, Tangye's special pump with steel pistons and rods is very handy.

**Leaching.**—The method of application of the several liquors is subject to considerable variation, according to the physical and chemical characteristics of the ore. The scheme embraces:—

(*a*) A preliminary wash, either with simple water or with an alkaline solution, to remove soluble salts: necessary not only with acid tailings, but also frequently with kiln-dried ore.

(*b*) Application of strong solution of KCy, with or without suction to hasten and perfect the drainage; and with or without transfer of the ore from one vat to another to effect aëration.

(*c*) Application of weak liquor as a wash to remove all traces of the strong liquor.

(*d*) Final water wash.

*Preliminary Wash.*—When the leaching tank has been properly filled and the surface has been levelled, the first wash is run on. In general practice, this alkaline wash is added to neutralise acidity and precipitate soluble acid salts, which are then supposed to be left in such condition as to be unacted on by cyanide. This, however, is by no means invariably the case. Iron, for example, which most commonly occurs, is present as ferrous and ferric salts, and the addition of the alkali wash precipitates these as ferrous and ferric hydrates. These hydrates are acted upon by cyanide * in the following manner:—

$$\begin{aligned}
6KCy &+ FeH_2O_2 &&= K_4FeCy_6 + 2KHO, \text{ and}\\
K_4FeCy_6 &+ FeH_2O_2 &&= K_2Fe_2Cy_6 + 2KHO\\
12KCy &+ Fe_2(H_2O_2)_3 &&= K_6Fe_2Cy_{12} + 6KHO, \text{ and}\\
K_6Fe_2Cy_{12} &+ 2FeH_2O_2 &&= K_2Fe_4Cy_{12} + 4KHO
\end{aligned}$$

The last equation is contrary to published statements, but it has been verified by James, who considers it possibly the reason why the addition of oxidising agents to cyanide solutions is "found in practice unnecessary and ineffective, an oxidising agent—viz. $K_6Fe_2Cy_{12}$ (+ 2KHO) being thus already present."

* A. James, Trans. Inst. M. & M., iii. 393 (1895).

A consideration of the reactions given above shows that it is necessary to precede the alkaline wash with a water wash, to free the charge from acid salts as fully as is economically possible. This water wash should be added from *below* the filter bottom; it thus rises through the charge, and passes over the side of the vat into a launder arranged for the purpose. It is then succeeded by an alkaline wash of either soda or lime, also added from below. By this means, acid liquor is prevented from contaminating the pipes and bottoms of the vats, which are thus always kept alkaline or neutral.

By a judicious discrimination in the use of alkaline agents, the solution may be preserved in an effective condition. A certain amount of alkalinity is advisable for successful economical working; on the other hand, an excess is to be carefully avoided. Lime forms insoluble compounds. Hence by employing this re-agent when the solutions are too alkaline, and soda when increased alkalinity would be of advantage, the balance may be assisted, and the solvent power of solutions be maintained. Dilute solutions of lime do not appear to be necessarily more destructive of cyanide than those of soda.

A common practice in New Zealand with very acid tailings, is to give a preliminary water wash and follow that with an alkaline wash carrying as much caustic soda in solution as is equal to 1 lb. per ton of ore, allowing it to stand for a couple of hours.

In Transvaal practice, "cyanicides" such as sulphuric acid, iron sulphates, and basic iron salts occurring in acid tailings, are remedied by adding caustic lime in a powdered form. With very acid tailings—i.e. pyritic tailings which have been exposed for some time to the oxidising influence of the atmosphere, as much as 2½ lb. per ton is added. With fresh tailings, ½ lb. of lime per ton is sufficient. In some works, the requisite quantity of lime is added to each car-load of tailings as it is hauled up to the leaching tanks. In others, 6 to 10 t. of tailings are dumped into the leaching tanks, and, after being levelled off, the lime is sprinkled over them. The practice of putting all the lime on the top of the tailings after the vat is filled cannot be recommended, as the lime forms a

pasty mass, and the alkalinity does not penetrate through the ore ; it will be found in deep tanks that the top layer, say for one third of the height of the ore, will be neutralised, whereas two thirds nearer the bottom will remain acid.

Sometimes an intermediate water wash is found advisable, to carry off the residues of alkali and alkaline products resulting from the previous wash, before adding KCy solution.

*Strong KCy Solution.*—In some works, sufficient "weak" solution—about 10 % of the amount of tailings in the leaching vat—is now pumped up from the sump on to the tailings, with the object of displacing the water left from the previous washing, and thus to prepare the way for the strong solution ; generally, however, this preliminary wash of weak solution is omitted.

The strong solution may contain ·3 to ·7 % KCy, and is usually about 33 % of the amount of ore in the charge. To prepare it, the foreman ascertains the strength of the solution in the strong sump, and also that of the liquor in the cyanide dissolver. He then commences to pump up the required quantity of solution from the sump to the upper reservoir, and, whilst this operation is being performed, runs in a calculated volume of liquor from the dissolver: this liquor is thus intimately mixed with the bulk on account of the agitation caused by pumping.

The amount of cyanide liquor required is calculated as follows :—Multiply the tons of solution taken by the difference between its present strength in cyanide and the normal strength required, and divide by the strength of the strong cyanide liquor less the strength of the normal solution required. This allows for the additional water introduced owing to the cyanide being in the form of liquor. Thus :—

Let T equal tons of solution taken.
,, A ,, present strength in cyanide of the sump solution.
,, B ,, the normal or required strength to which T is to be brought.
,, C ,, strength of liquor in cyanide dissolver.

Then $\frac{T(B - A)}{C - B}$ is the amount (in tons) of liquor to be added.

If it is required to make a solution of 30 t. of ·5 % KCy, the strength of solution in the strong sump being ·35 %, and of the liquor in the cyanide dissolver 20 %, then—

$$\frac{30(\cdot 5 - \cdot 35)}{20 - \cdot 5} = \frac{30(\cdot 15)}{19\cdot 5} = \frac{4\cdot 5}{19\cdot 5} = \cdot 231 \text{ t., or } 46\cdot 2 \text{ gal.}$$

The solution now in the reservoir will be 46·2 gal. more than 30 t., but this slight excess is not material, the strength in cyanide of the total amount being correct.

The strong solution is run from the reservoir on to the top of the charge. It was formerly the practice to allow the solution to stand for 12 hours or so in contact with the charge before running off from below the filter, but very generally it is allowed to come away at once, and the cocks or clamps on the exit pipe are so adjusted that the solution may flow evenly during the whole 24 hours or so of contact. By the first method, the 12 hours' contact is maintained with only that portion of the solution which is diluted with the moisture already present (except in the case of dry-crushed ores), the cyanide of which has been probably largely spent by the first contact with the contents of the charge during its passage through the mass; the solution is thus in the most likely condition for accidental decomposition at the bottom of the filter vat. By the second method, fresh portions of solution are being continuously applied to each particle of the charge. In some cases, it is found to be of advantage to allow solutions to percolate more rapidly, and to pump them back on to the top of the charge, and thus to "circulate" them.

When the ores contain much silver, or if concentrates are being treated, it is sometimes desirable to increase the amount of solution from 1 : 3 to 1 : 2, or even 1 : 1 of the ore; but generally it results that increased proportions of solution show an increased consumption of cyanide.

Vacuum suction is not applied to the strong solution leach, except in very stubborn cases, and also where agitators

have been used, as it is found that the percolation rate has usually to be retarded rather than accelerated. In wet weather, however, when tailings are charged into the vats in a sloppy condition, percolation is hindered, and suction may be required. Under these conditions, it is beneficial, where practicable, to allow longer time for leaching and treatment generally, or the extraction percentage may be temporarily lowered.

*Weak KCy Liquor.*—The "weak" solution is run on as soon as the top of the charge appears to be dry. Sometimes the ore is allowed to drain quite dry, to permit access of air, but this is usually avoided ; in leaching some ore, however, turning over the charge by hand between the various treatments of solution, added in instalments, has a distinctly beneficial effect. Weak solution is usually ·05 to ·15 %, and the quantity taken varies from one-half to double the amount of the strong solution used. The amount required is determined by preliminary investigation ; formerly it was seldom more than the strong solution, and was regarded rather as a washing to free the charge of as much strong solution as possible before the final water washing, but it is now considered to play an important part in the solution of the gold, and receives more attention accordingly. The rate of percolation of this solution is usually increased by the application of the vacuum apparatus.

The strengths of strong and weak solutions given here are those most used in actual practice. It must, however, be understood (on the authority of Alfred James) that "strong" solutions of ·1 % and under are sometimes effective, followed by correspondingly dilute "weak" solutions ; but the increased time required to obtain efficient extraction has generally resulted in the adoption of the more rapidly effectual strengths given. Increased liability to decomposition and consequent re-precipitation of the gold has been noticed in one case where extremely dilute solutions were used.

*Final Water Wash.*—Wash waters are added to the top of the charge, and made to percolate as rapidly as possible, by the aid of the vacuum apparatus where convenient. The

amount of water taken varies with the quantity of solutions circulating in the works. If the works are full, the last washing is usually given with weak solution. This practice is not to be recommended, as it causes a loss both of gold and of cyanide, which leaves the works in the "moisture" (15 to 20 %) contained in the residues when discharged. It is preferable to have an extra precipitator, through which the water washings may be run. This saves the gold, and permits the stronger portion of the cyanide solutions coming off to be also retained. A water washing should be about one-third of the weight of the charge, to do its work effectively; but in practice, in order to maintain the "balance" of solutions, it is frequently only 5 or 10 %.

*Examples.*—(i.) The following summary of time occupied in washing and leaching at the Rand Central works is quoted by Eissler :—

| | | | |
|---|---|---|---|
| (*a*) | Alkaline wash: | filling . . . . . . . . | 2 hours. |
| | | leaching . . . . . . . | 3 ,, |
| (*b*) | Strong solution: | filling and contact . . . | 5 ,, |
| | | leaching . . . . . . . | 8 ,, |
| | | draining dry . . . . . | 4 ,, |
| (*c*) | Weak solution: | 4 washes, 1 hr. ea. . . . | 4 ,, |
| | | 4 leachings, 4 hr. ea. . . | 16 ,, |
| (*d*) | Water wash: | filling . . . . . . . . . | 1 ,, |
| | | leaching . . . . . . . . | 7 ,, |
| | | Total time . . . . | 50 hours. |

This treatment is for $7\frac{1}{2}$ %-dwt. tailings. The alkaline wash contains ·16 % KCy and 4 oz. caustic soda per ton of solution; the strong liquor has ·35 % KCy; the consumption of KCy is $\frac{3}{4}$ lb. per ton of ore; and the residues assay 15 gr. to 1 dwt.

(ii.) The system of "direct filling" adopted at the New Kleinfontein * requires 2 slime separators (pointed boxes, 6 ft. sq. at top and 6 ft. deep), 19 leaching tanks (13 of 200 t. capacity and 6 of 130 t.), and 9 solution tanks (total capacity 636 t.). About 15 % of slimes are eliminated, and the remaining pulp is run through launders into the leaching vats, which are previously filled with water. In the course of filling the vat, another 10 % of slimes overflows into the slimes

* F. Cardell Pengilly, Trans. Inst. M. & M., vi. 113 (1898).

VAT A.: CAPACITY 207 T. CHARGE I.

| Date. | Time. | Test. | — | KCy solution t. |
|---|---|---|---|---|
| Tue. 5 | 1 p.m. | — | Started filling . . . . . . . . . | — |
| Wed. 6 | 2 a.m. | — | Full; draining dry . . . . . . . . | — |
| ,, | 4 a.m. | acid | Added alkaline wash; 20 lb. NaHO and ½ bag lime . . . . . . . . . | — |
| ,, | 9 a.m. | alkaline | Prelim. solution ·16 % KCy . . . . | 20 |
| ,, | 2 p.m. | ·01 | Gold solution to weak precipitating box . | — |
| ,, | ,, | — | First strong solution, ·3 % KCy . . . | 30 |
| ,, | 3 p.m. | — | do. . . . | 20 |
| ,, | 7 p.m. | — | do. . . . | 6 |
| ,, | 9 p.m. | ·09 | Standing under solution . . . . . . | — |
| ,, | 12 p.m. | — | Started leaching . . . . . . . . . | — |
| Thurs. 7 | 1 a.m. | ·10 | Gold solution to medium box . . . . . | — |
| ,, | 3 a.m. | ·13 | — — | — |
| ,, | 6 a.m. | ·16 | — — | — |
| ,, | 9 a.m. | ·18 | — — | — |
| ,, | 12 noon | ·20 | Gold solution to strong box . . . . . | — |
| ,, | ,, | — | Second strong solution, ·3 % KCy . . | 30 |
| ,, | 2 p.m. | — | do. do. . | 10 |
| ,, | 6 p.m. | — | do. do. . . | 15 |
| ,, | 8 p.m. | — | do. do. . . | 1 |
| ,, | 8 p.m. | ·28 | Standing under solution . . . . . . | — |
| ,, | 11 p.m. | — | Started leaching . . . . . . . . . | — |
| Fri. 8 | 8 a.m. | — | Finished leaching . . . . . . . . | — |
| ,, | ,, | — | Medium solution, ·26 % KCy. . . . . | 20 |
| ,, | 12 noon | — | do. . . . . . | 5 |
| ,, | 4 p.m. | — | do. . . . . . | 8 |
| ,, | 8 p.m. | — | do. . . . . . | 3 |
| ,, | 9 p.m. | ·20 | Gold solution to medium box . . . . . | — |
| ,, | 12 p.m. | — | Medium solution . . . . . . . . . | 8 |
| Sat. 9 | 5 a.m. | — | do. . . . . . . . . . | 10 |
| ,, | 12 noon | — | do. . . . . . . . . . | 7 |
| ,, | 3 p.m. | ·18 | — — | — |
| ,, | 4 p.m. | — | do. . . . . . . . . . | 10 |
| ,, | 8 p.m. | — | do. . . . . . . . . . | 7 |
| Sun. 10 | 4 a.m. | — | do. . . . . . . . . . | 12 |
| ,, | 8 a.m. | — | do. . . . . . . . . . | 5 |
| ,, | 6 p.m. | — | Weak solution, ·14 % KCy . . . . . | 5 |
| ,, | 12 p.m. | — | do. . . . . . | 5 |
| Mon. 11 | 6 a.m. | — | do. . . . . . | 5 |
| ,, | ,, | — | Gold solution to weak box . . . . . | — |
| ,, | 12 noon | — | Weak solution . . . . . . . . . . | 5 |
| ,, | 6 p.m. | ·09 | do. . . . . . . . . . . | 5 |
| ,, | 7 p.m. | — | do. . . . . . . . . . . | 5 |
| ,, | 9 p.m. | — | do. . . . . . . . . . . | 5 |
| ,, | 12 p.m. | — | do. . . . . . . . . . . | 4 |
| Tue. 12 | 6 a.m. | — | Draining dry . . . . . . . . . . | — |
| Wed. 13 | 6 a.m. | — | Discharging . . . . . . . . . . . | — |
| | | | Total KCy solution . . . . . | 266 |

race, so that the resulting tailings to be treated in the vat contain but a small proportion of slimes. The vat is filled with tailings (sand) to within 1 ft. of the top. Each vat, after being drained of water, is treated with an alkaline solution. As soon as the solution draining away is slightly alkaline, the treatment by KCy liquor is commenced. Various strengths of solution are pumped into the vat, each vat receiving during its course of treatment 200 to 275 t. of cyanide solution. The length of treatment is 6 days, and the amount of KCy used is ·6 lb. per ton of tailings. Opposite is a page from the log book of the works, which shows the exact method of procedure adopted.*

(iii.) At the Luipaard's Vlei mine, Transvaal, over 20,000 t. of Rand banket have been treated by Franklin White† on the "interrupted" lines suggested by W. R. Feldtmann, which involves the use of two sets of tanks, so that after three courses of liquor treatment, the ore can be transferred to separate tanks before repetition. The object of this is to secure thorough aëration, but White thinks, though it may be advantageous with wet-crushed tailings, it is not so with dry-crushed ore, which has abundance of entangled air, but rather that the transference of damp ore to the second tanks causes packing and hinders filtration; it seems probable, however, that when a vacuum is created, the pulp can be transferred in a much drier condition, and this evil be minimised. No doubt the bottom 2 or 3 ft. of pulp in a deep tank is often less well treated than the bulk, and by the transference method it would get on top next time.

The crushed ore (see p. 311), having had 1 to $2\frac{1}{2}$ lb. lime per ton added at the start, is filled into steel tanks 25 ft. diam. and 8 ft. deep, with the usual filter bottoms and discharge doors. As soon as a tank in the first row is filled, a strong solution (·25 % KCy) followed by two others, the last being

* This "log book," as printed in the Trans. Inst. M. & M., vi. 115, is somewhat of a curiosity. According to it, "1 p.m." is an earlier hour than "2 a.m." on the same day, and one day has a "12 p.m." twice; moreover, the "tons of KCy solution" do not add. Without considerable amendment, it reads absolute nonsense.

† Trans. Inst. M. & M., vii. 124 (1899).

·15 % is pumped on to it. The time given to this treatment is 66 to 70 hours. The solutions are not allowed to stand, but are drained off when the tank is filled, the object being to allow fresh air to obtain access to the mass of damp sand. Each solution is about 27 t. to the tank of ore (165 to 170 t.).

The ore is then transferred to the second row of tanks, having already lost about 67 % of its original assay value. Probably the abundance of air entangled in the dry sand, as compared with what would be held in a tank of sand settled with water, materially assists the solution of the gold, while the finest grains of the free gold would be quickly taken up. A solution of 20 to 25 t., not exceeding ·2 % KCy, is then pumped on to the transferred sands, and drained off, the mass being allowed to remain damp for about 96 hours, when weaker solutions (·15 to ·1 %) are used in continuous washes, making up a total of 75 t. per tank (2nd treatment). A water wash of 20 or 30 t. completes this part of the process, which would last some 275 hours.

It is worthy of note that for the sake of an additional 1 dwt. or so extraction, no less than 11 days' further treatment is required, and it would seem that the time consumed, wages incurred in transference, and great extra plant needed, form together a somewhat cumbrous and costly scheme, having for sole object the better aëration of the pulp.

A careful series of moisture tests and measurements of solution sumps during the treatment of 4 tanks (680 t.) showed that the total loss of liquid in the treatment was 96 t., or 24 t. per tank. The moisture in the discharged residues averaged 12·3 %, or, say, 20 t. per tank; the remaining 4 t. would be represented by evaporation from surfaces of sumps and tanks and by leakages. As the ore contained about 3 t. of water per tank, in the form of moisture, when delivered from the mill storage bin, the actual consumption of fresh liquid equals 21 t. per tank, or $\frac{1}{8}$ t. (25 gal.) per ton of ore.

The solutions running from the first tanks carried 13 to 32 dwt. per ton; those from the second, 3 to 4 dwt.; and the final wash, ·8 to 2·5 dwt.

During the treatment of 22,500 t. or say 130 tanks (repre-

senting solutions and washes equal to about 23,500 t.), the solutions had been repeatedly used over again, and it was thought they might be impaired by destruction of their oxygen contents, but all were found to be in working order. The mine water used for dissolving the cyanide contained 1·33 mgm. of oxygen per litre; strong KCy liquor, 3·45 before use, nil as it commenced filtering off, and 2·8 at end of filtering; weak KCy liquor, mechanically aërated, 6·4 mgm. (saturation) going on, and 6 mgm. coming off.

Tests made for lime and acidity showed that somewhat "more than 1 lb. lime per ton of ore was required during the whole treatment."

**Treating Pyritic Ore.**—With concentrates, and with sulphuretted or telluride ore, in the raw state, the treatment with ordinary KCy liquor needs to be much more prolonged, ranging from a fortnight to a month.

In the Transvaal, commonly 16 to 18 days are occupied on vanner and box concentrates. The solutions used are ·25 %, ·1, and ·05 %, with about 1 lb. lime per ton. The strong solution is run on for 4 days or more, till it leaves the tank at the same strength as it entered; the weak solutions are constantly circulated till a high point of extraction is reached. This is often 94 %, the residues assaying $1\frac{1}{2}$ dwt. KCy consumption is only about $1\frac{1}{2}$ lb. per ton. Sometimes sand is added to keep the mass from packing so densely and thus to facilitate percolation.

Cyanide treatment of vanner concentrates, with agitation, was tried by S. J. McCormick * at Ouro Preto, Brazil, using $1\frac{1}{4}$ % KCy solution in revolving barrels driven by a Pelton wheel, 12 cwt. concentrates being charged in with 1000 lb. KCy liquor, and rotation being maintained for about 40 hours. At the end of that time, the gold-cyanide solution was still strong in KCy, and was conserved for further application. The first wash waters were also preserved, and being strong in KCy were used to make up strong liquor to the required amount. After treating 4 charges, the solution was sent to

* Trans. Inst. M. & M., v. 120 (1897).

the extractors and pumped back to the stock tank, still carrying ·3 % KCy and a little gold. The consumption of KCy was 11 lb. per ton, and the extraction was 80 to 85 %. By percolation in a cement-lined brick tank, the process occupied a month, and consumed 13¼ lb. KCy per ton; extraction reached 85 to 90 %. As compared with chlorination, the recovery was much less but the cost was also less, owing to the heavy (local) item for fuel in the former case. The comparative results on 100 t. assaying 2·88 oz. gold are given as follows :—

| | | Chlorination. | | KCy agitation. | | KCy percolation. | |
|---|---|---|---|---|---|---|---|
| | | *s.* | *d.* | *s.* | *d.* | *s.* | *d.* |
| Cost: | Labour | 11 | 2 | 4 | 4 | 1 | 8 |
| | Chemicals | 4 | 9 | 16 | 10 | 20 | 5 |
| | Fuel | 10 | | | | | |
| | Repairs | 0 | 6 | | | | |
| | Total | 26 | 5 | 21 | 2 | 22 | 1 |
| | Extraction : | 92 % | | 85 % | | 85 % | |

The highly pyritic tailings produced in mills operating on the Black Reef of the Rand can be dealt with without liming when treated fresh from the battery, but their consumption of KCy is high. On 11-dwt. material using ·5 % solution, the extraction is 10·2 dwt., and the cyanide consumption is 3·8 per lb. per ton; with ·3 % liquor the extraction is 9·3 dwt., and the consumption is 2·75 lb.

Frequently in giving heavy sulphide ores lengthened percolation, there is considerable difficulty, as, after the first few days, the ore settles down hard in the vat, and renders filtration and washing slow and imperfect. With sulphide ores, in which the gold is difficult to dissolve, and which require either lengthened exposure to the solution or brisk agitation, the following continuous method of treatment might be successfully applied :—Crush the ore with cyanide solution and pass the pulp over plates; then run into a series of agitators, so arranged that the pulp will overflow continuously from the one to the other until each particle in passing through will have the necessary agitation for its treatment; and finally convey the overflowing pulp from the last agitator into a vat, where it can be filtered and washed.

*Oxygenating.*—Elsewhere difficulty has been encountered in cyaniding pyritic ores, mainly due, it is said, to the rapid exhaustion of the supply of oxygen; and the use of various "oxygen-carriers" has been proposed. In some cases, the initial dissolving action has been much intensified; but in others, the consumption of both oxidant and cyanide has been excessive. On this subject, H. L. Sulman remarks* that "the fatal defects in all air oxygenation—or chemical oxidation—cyanide processes are (1) the oxidation of iron and other base sulphides, with consequent direct destruction of cyanide; and (2) the production of caustic potash as a necessary reaction-product of the solution of gold. With free-milling ores, containing none of the worse "cyanicides," the latter is not a great drawback; but directly sulphides of iron, copper, arsenic, antimony, etc., make their appearance in an ore, soluble compounds are produced from them (partly by reason of the caustic alkali formed), which act directly as cyanide destroyers."

Any wholesale condemnation of oxygenation processes, however, is opposed to the results which have been actually attained by them in many quarters, notably by the aid of sodium dioxide.

In the United States, the addition of sodium dioxide to the KCy solution is quite common, and is reported to give beneficial results at many works.

Australian operators, in Victoria and in New South Wales, have for some years been paying royalty for the use of J. C. Montgomerie's "binoxide of sodium," notwithstanding that the reagent costs 4*s.* a lb. and that about $\frac{1}{4}$ lb. per ton is required. The rapidity and efficiency given to the solvent action of the KCy liquor by its use are regarded as sufficient to repay the cost. The salt ($Na_2O_2$) decomposes very rapidly in water, its nascent oxygen combining with the potash of the KCy and liberating the cyanide. The extraction obtained is not said to be any greater than with ordinary cyanide solution, and the consumption of KCy is actually greater, so that the only benefit derived is increased rapidity, permitting

* Trans. Inst. M. & M., iii. 204 (1895).

additional output for a plant of certain size, and thus effecting a small saving in capital outlay; but it can hardly be economical in the end. The disposition to sacrifice economy for the sake of speed is rather characteristic of American practice, and is noticeable in their chlorination mills also.

*Bromo-Cyanogen.*—With ores of this class, greatly increased rapidity of dissolution of the gold, with diminished cost for solvent, is claimed by Sulman and Teed to be secured by their bromo-cyanogen solution. These results are ascribed mainly to the enormously accelerated solution of the gold, wherever its particles are easily accessible, by having "potential cyanogen" always ready at hand to effect the formation of the aurous cyanide necessary for the soluble double salt of gold and potassium; and secondly to the avoidance of production of caustic potash—the product of their reaction being quite inert as regards the further formation of cyanide-destroying compounds.

Instead of effecting the "potentiality" of cyanogen by adding oxygen, they use a halogen.

When dissolved oxidants are used, such as hydrogen peroxide, etc., the tendency is to oxidise other ore compounds also—the iron pyrites, copper sulphide, etc.—with fatal results to the potassium cyanide. In the case of the latter mineral, and in those of arsenical and antimonial compounds, the production of "cyanicides" is intensified by the caustic potash produced in the solvent reaction giving rise to soluble sulphides. The quantity of KHO liberated is of course small, but as it is formed exactly in proportion to the gold dissolved, and at the actual point where such solution is proceeding, it is in precisely the position where it can produce the most harmful results.

Many of the cyanicides generated are powerful oxygen absorbents, and intercept the supply required for dissolution of the gold. With the halo-cyanogen solvents, however, the "potential store of that radicle is always molecularly present in the required proportions," while the halogen compounds themselves are said to be inert with regard to most metallic sulphides.

Cyanogen bromide is a white, crystalline, salt-like compound, having a strong penetrating odour, and is solid within all ordinary ranges of air temperature. It is slowly volatile on exposure, but does not evaporate from dilute aqueous solutions to any marked degree. Beyond its pungency and slight volatility, it presents no difficulty in handling. It may be preserved unaltered for indefinite periods when stored in well closed vessels, and can be exported in tins which have previously received an interior varnish of paraffin wax.

If cyanogen bromide be added in definite proportion to a solution of potassium cyanide, no change takes place in the chemical nature of the compounds present, the solution remaining stable for an indefinite time. In practice, solid cyanogen bromide is added to weak KCy solution in a proportion not exceeding, chemically, 1 molecule of the bromide to 3 of cyanide, or in parts by weight, about 1 to 2. This, however, is a maximum proportion never to be exceeded, and it is preferable to use less of the bromide than is required for chemical inter-action with the whole of the cyanide present, for the simple reason that in a leaching operation the whole of the cyanide is not required to be used up in the gold extraction. In other words, the amount of cyanogen bromide should be apportioned to the work to be done, according to preliminary trials. If the strength of say ·6 % KCy + ·3 % BrCy be exceeded, there is a tendency to decomposition of the joint solvent, with liberation of brown paracyanogen. Such a strength, however, would never be necessary for a solvent, and the more dilute the liquors become the greater apparently is their stability. As in the case of ordinary cyanide, the tendency of latest extractions has been towards the use of weaker solutions.

Cyanogen bromide being unaffected by silver nitrate, the free potassic cyanide may be titrated with a standard silver solution. The cyanogen bromide existing in the liquor may be estimated by rendering acid with hydrochloric acid, adding slight excess of potassium iodide solution, and titrating the resulting liquid against standard sodium thiosulphate, with a

starch indicator as usual: 1 c.c. of $\frac{N}{10}$ thiosulphate corresponds to ·0053 grm. cyanogen bromide.

Generally speaking, a leaching of 24 hours has been sufficient even for large masses of complex and pyritic concentrates, and in every case 48 hours has shown the maximum extraction possible.

Air being necessarily present in all cyaniding operations, it is admitted that a small amount of caustic potash may be formed even in the bromo-cyanogen process; but preference is given to closed vats, and air is excluded as much as possible.

The foregoing remarks are based on statements by the inventors of the bromo-cyanogen process.

Prof. A. K. Huntington finds that a good deal of the cyanogen bromide undergoes decomposition in 3 hours when brought into contact either with gold ore containing iron oxide, or with an ordinary iron ore, though he would not imply that the cyanogen bromide will not have done a good deal of work in that time, nor that there will not be so much of it left as may suffice to carry on the work to the end. Clearly, however, the point raised by Prof. Huntington is found in practice to be one of considerable importance in its bearing on the commercial aspect of the process, for at the works in W. Australia, where it is being put in operation, a special kind of mill is used (see p. 301) for fine crushing, the object of which is to prevent iron (abraded from the ordinary steel balls) from getting into the ore.

The first notable instance of the bromo-cyanide process being actually installed on a working scale was at the Canada Consolidated Co.'s mill, Deloro, Canada, where it replaced the barrel chlorination of roasted mispickel ores. The plant and operations have been described at length by H. F. K. Picard,* who was in charge.

The ore is dry crushed in ball mills and similar machinery to 30 mesh, and from it is removed by air-separation 12 % of untreatable and practically valueless (under 2 dwt.) slimes.

* Trans. Fed. Inst Min. Engs., xv. 417 (1898).

The sands are trucked to 4 circular leaching vats 18 ft. diam. × 5 ft. deep, capacity 40 to 50 t., made of 3-in. staves of Californian redwood. Each charge is treated with 15 t. of strong liquor, holding ·18 % KCy and about 5 lb. of cyanogen bromide previously dissolved in 5 gal. water, further instalments of bromide being added at intervals to make up a total of 15 to 20 lb. per charge, dependent on the prevalence of sulphides in the ore. Precautions for minimising escape of pungent fumes when handling the bromide solution are necessary.

No addition of lime or soda is made to the ore to correct acidity, nor is any preliminary wash used.

Leaching is much hastened and gold-dissolution is materially increased by causing a circulation of the liquor by means of a centrifugal pump, drawing it away at the bottom of the vat and returning it to the top again. The operation continues for 36 to 48 hours. It is followed by 10 to 14 t. of weak liquor (·05 to ·08 %), which is regarded as more than is really necessary; and this again by the usual water wash. The gold-charged liquor is used repeatedly on fresh ore, so that it becomes quite rich before precipitation.

Fig. 162 shows the general arrangement of the plant: *a*, dissolving tank for KCy and BrCy; *b*, leaching tanks; *c*, centrifugal pump for agitating pulp with solutions; *d*, truck for solid residues; *e*, zinc-fume emulsion tank; *f*, cone precipitators; *g*, store tanks.

From ore assaying 15·62 dwt. per ton on the average, an extraction of 13·28 dwt. was got, leaving 2·34 dwt. in the residues. This was accomplished with a consumption of ·8 lb. KCy and ·4 lb. bromo-cyanide per ton, the total cost being given at 5*s.* 7*d.* per ton when treating 600 t. per month. With a larger installation it should be lessened.

Since 1896, bromo-cyaniding has been most successfully practised at the Carolina mill of the Argentine Concessions Limited. Here the material operated on is pyritic schist, and an extraction of 95 % is reported. The process replaced pan amalgamation and roasting, which involved much labour and fuel. In this instance, fuel was a very heavy item, the

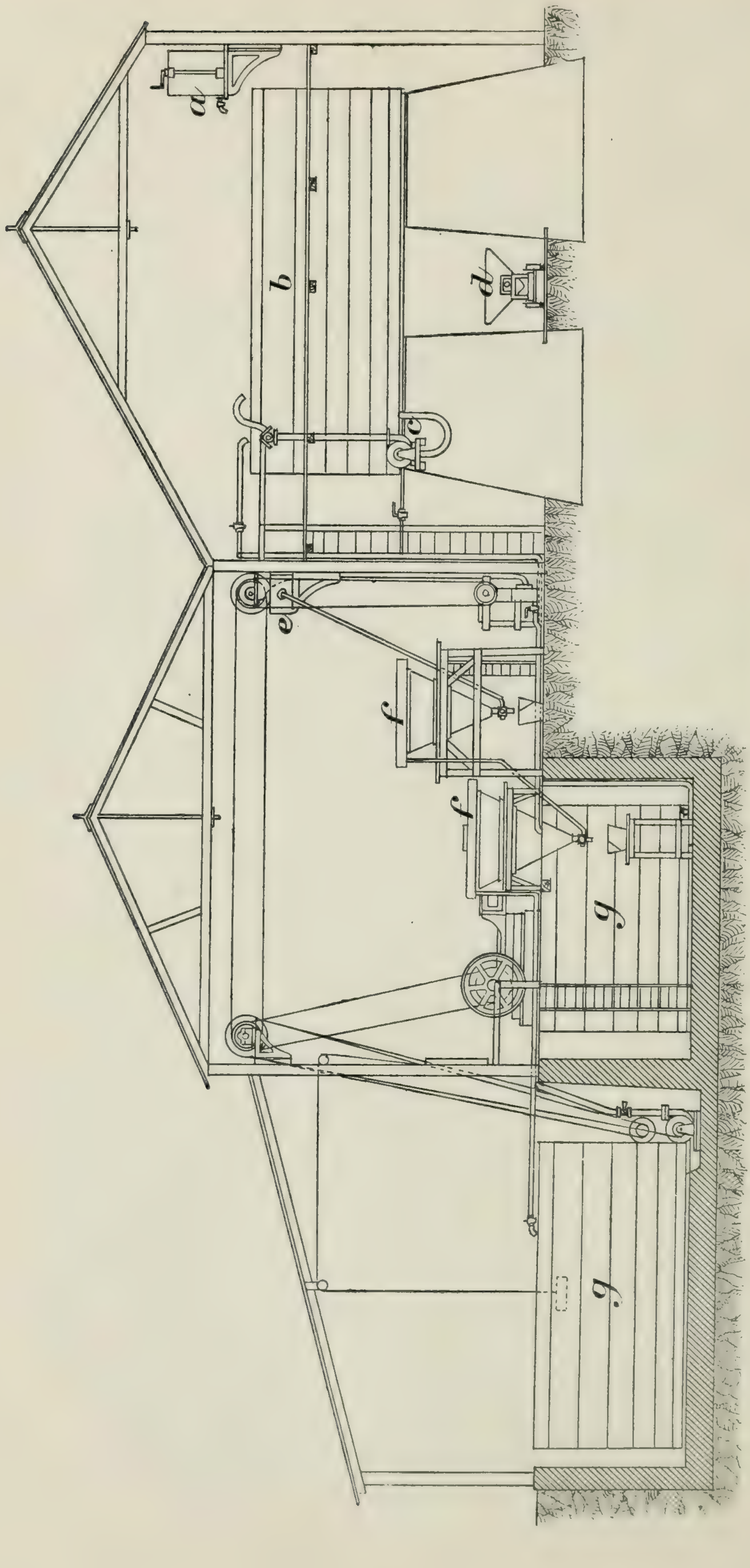



FIG. 162.—BROMO-CYANIDATION PLANT.

works lying above the timber zone; but wages for common labour rule low.

Under the name of the Diehl process, bromo-cyanide is undergoing trial at several plants in Western Australia, notably at the Lake View Consols and at Hannan's Brownhill, a minimum extraction of 90 % at a maximum cost of 30*s.* per ton being, it is said, guaranteed, while the royalty payable is 2½ %. This is really the Sulman-Teed process, Dr. Diehl being the local representative of the Hamburg firm who purchased the West Australian rights.

In June 1900 it was reported that a trial run at Hannan's Brownhill on 635 t. had given an actual recovery of 95·75 % at a cost of 28*s.* 3*d.* per ton, including smelting of concentrates.

The treatment embraces wet crushing (very fine), amalgamation, concentration of mineral, sizing of pulp (all sands being returned for re-grinding to slimes), agitation, cyaniding (plus bromo-cyanide), and filter-pressing.

In experimental treatment of Hannan's Star ore (see analysis p. 489), James obtained the following comparative extractions* :—

| Ore assay. Oz. per ton. | Mesh. | KCy. % | BrCy. % | Period. Hrs. | Extraction. % | Residues assay. Oz. per ton. |
|---|---|---|---|---|---|---|
| 6½ | 120 | ·4 | nil | 92 | 68 | 2·00 |
| 6½ | 120 | ·4 | ·04 | 23 | 91½ | ·55 |
| 6½ | 40 | ·4 | nil | 69 | 65 | — |
| 6½ | 40 | ·4 | ·04 | 69 | 78 | — |

The consumption of KCy was apparently somewhat lessened when bromo-cyanide was used, due doubtless to the shortened period of treatment; but against this is to be placed the cost of the bromine salt, and the royalty charges. It is not made clear how the separated rich concentrates are to be dealt with, what their proportion to the whole will be, what percental extraction will be got from them, nor at what cost.

* Trans. Inst. M. & M., viii. 486 (1900).

James thinks the 90 % extraction guaranteed may be possible without reckoning on the concentrate contents, if reduction is carried to 120 mesh, but he doubts whether the tailings can be brought as low as by roasting, though the elimination of the most refractory constituent (at least in part) by concentration may materially simplify the treatment of and enhance the gold recovery from the remainder.

Thus bromo-cyaniding has been established successfully in widely distant fields, and to deal with three distinct classes of pyritic ore—mispickel, common pyrites, and tellurides.

*Roasting.*—This operation has been described in detail in a previous chapter (pp. 361–409).

Many trials have been made of its effect on cyanide treatment for the Kalgurli sulphides, which carry about ·03 % tellurium.

On a laboratory scale, James got an extraction of 60 to 77 % from raw ore by 16½ hours' agitation ; and after roasting the residues from the 60 % experiment, he obtained a further recovery bringing the total to 93 %. Roasting caused a loss in weight of about 4 %, and brought the extraction by ordinary agitation for 16½ hours from 60 % to 87 %.

With three samples ground to pass 60 mesh and treated with ·3 % strong solution of KCy the extractions were as follows :—

| Ore. | Solution. | Extraction. % | Days |
|---|---|---|---|
| Raw . . . . . . . | ·3 % KCy | 80 | 25 |
| Roasted . . . . . | do. | 93 | 29 |
| Raw . . . . . . . | KCy & BrCy | 93½ | 24 |
| Roasted . . . . . | do. | 85 | 24 |

In these tests it was noticed that the roasted ores contained a large percentage of soluble salts, and that there was a heavier consumption of cyanide on the roasted ore, showing the necessity for some intermediate treatment between roasting and cyaniding to get rid of the soluble salts. Blowing air

into vats under the filter bottoms, and into the solutions, did not accelerate or improve the extraction on either raw or roasted ore, provided the solutions were circulated with ordinary rapidity.

It is only natural that comparisons should be drawn between methods of telluride treatment adopted in Colorado and their application in Western Australia, but the conditions are totally different. All the Cripple Creek ores repay smelting or roasting, and offer no greater difficulty than ordinary sulphurets (see p. 368); but very large quantities of the West Australian ore do not assay above 5 to 15 dwt., and such cannot bear roasting costs, besides which the lime in the gangue precludes the adoption of chlorination with its high rate of extraction. Moreover, it would seem that there is always a risk of the tellurides being more or less segregated into small patches, even in the poorest of the Westralian ores, and these would on roasting be liable to leave the gold in a condition not easily attacked by any solvent, necessitating some intermediate method for its recovery. Again, at Kalgurli fuel is dearer and less suitable than that obtainable in Colorado.

Nevertheless roasting and simple cyaniding are favourably regarded in some quarters, and have been carried to a successful issue at least in one instance, the Great Boulder Main Reef.

Since January 1900 * this plant has been working, sometimes on roasted sulphides and sometimes on raw tailings (sands and slimes). With 5-oz. sulphide ore, the extraction is said to be 97 % (equivalent to 3 dwt. loss in residues) at a cost of 19*s.* per ton, including 4*s.* per ton for roasting. Treating raw tailings, 726 t. yielded 558 oz., being 93·18 % of the contents, and 2000 t. of lower-grade tailings gave 90½ %, at a total cost (no crushing or roasting) of 12*s.* per ton. The practice embraces pan-grinding and amalgamation to catch the coarse and free gold ; removal of excess of water (carrying with it in solution those salts which cause a loss of cyanide) by means of filter-presses ; agitation-cyaniding with

* Official Report, May 9, 1900.

clean water for KCy solutions ; and filter-pressing to separate residues. Roasting is only used when the percentage of sulphurets compels it. The consumption of KCy is given at 8 to 9 lb. per ton, and the cost at 10*s.* Dry crushing (breakers and ball mills) is employed, and the pans are arranged in series, so that the ore passes automatically from one stage to another.

Similar procedure is under consideration or already adopted at the Ivanhoe, Golden Horseshoe, and South Kalgurli mills, and it is said that the washing out of soluble salts by the preliminary filter-press treatment has brought down cyanide consumption to 1½ to 2 lb. per ton, while the residues assay only 1 to 2 dwt. But roasting and filter-pressing are both costly proceedings, and the slaking of some ores after roasting would result in formation of a cemented mass.

**Treating Slimes.**—Not only does the percentage of slimes to sands vary in almost every example of mill tailings, but there is also the widest possible discrepancy in the distribution of the gold. Thus in some cases, though admittedly rare, when tailings have been subjected to hydraulic classification, so much of the value is removed in the coarse grains of ore and heavy grains of pyrites that the remaining slimes are practically worthless, or contain so little bullion as not to repay treatment, according to our present knowledge, and are therefore run to waste. On the other hand, there are very many cases where the isolated slimes possess ton for ton as much value as the sands ; and even quite a number in which the slimes are by far the richer portion. Thus it is frequently a matter of prime importance to successfully and economically treat the slimes.

But it was very soon discovered in practice, as might have been inferred from their physical character, that no ordinary leaching process would be of the least utility, because percolation is out of the question. Numerous methods have been proposed for avoiding the difficulty. Where the proportion of slimes is quite small, the ore hard and silicious, and the climate hot, as in the case of India, it has been found

practicable to mix the sun-dried slimes with a percentage of coarse clean sands, and thus to carry out ordinary leaching with the aid of vacuum suction for accelerating and improving the final washings. But such conditions exist only in very limited areas, and such treatment does not affect the main proposition.

There are two principal methods to choose from. In one, the coagulation of slimes is sought, so as to admit of the application of leaching by percolation ; in the other, slimes and solutions are agitated together, and then separated either by deposition or by filtration in some form.

*Coagulation.*—It has been found by several experimenters that the addition of milk of lime, or lime-water, to slime pulp will bring about such a coalescence or flocculation of the particles that leaching is sometimes rendered feasible, or, as is more common, that, after agitation and settlement, it is a comparatively easy matter to separate slime solids from solution by "decantation" or siphoning off the liquor, as described further on (p. 577). But it depends much upon the ore.

With a calcareous gangue, addition of lime often increases the time occupied in settling.

It was found with the refractory and slimy ore of the Republic mine, Washington (see analysis p. 487)* that treatment with sulphuric acid caused some of the suspended material to go into solution while the rest began to subside, and on making the mass alkaline again, flocculation and subsidence ensued. Without previous acidulation, neither lime nor salt had any appreciable effect in causing settlement. In this case, hydrated oxides caused the trouble ; but with limey pulp the effect might be equally good, an insoluble sulphate being formed.

In the case of weathered ores yielding ferric hydrate, deposition is much slower than with freshly milled ore.†

It has been found by H. L. Sulman‡ that by causing the formation of a uniform small amount of a coagulum throughout slimes, the whole of the slimes, together with any trace

* En. & Min. Jl., April 28, 1900.

† W. A. Caldecott, Jl. Chem. & Met. Soc. S. Africa, i. 96 (1898).

‡ Trans. Inst. M. & M., iii. 226 (1895).

of gold, are precipitated with great rapidity, and that this precipitate is far more amenable to leaching than the original slimes. The most efficacious coagulating agent proved to be that produced by addition of a soluble metallic or alkaline-earth salt to a dilute solution of soap. The effect produced by the addition of a little milk of lime to the dilute soap solution containing suspended slimes is very marked. The small amount of sticky and contractile calcium oleate and stearate formed throughout the mass instantaneously sweeps together, and binds into more or less granular flocky masses, the most finely divided particles hitherto in suspension. The physical condition of the precipitate produced by means of this coagulation method is entirely altered, the nature of the deposit being very different from that of ordinary slimes. The amount of soap required is very small, being only 2 to 5 lb. per ton of suspended slimes—reckoning the latter as dry material. The precipitated slimes, instead of packing together to form a layer impervious to the passage of water, now admit of the passage through them of leaching solutions.

Thus it only becomes necessary, according to Sulman, to "collect in a shallow tank the effluent slimes from the hydraulic classifiers, suspended in a very dilute soap solution, and to add thereto a small amount of milk of lime, in order to effect the complete precipitation of all slimes and all suspended matter (however finely divided) into one uniform precipitate, in a leachable condition. The clear effluent, containing neither soap, slimes, nor any trace of gold, may be run away, and the slime precipitate be transferred to a suitable leaching vessel. By the adoption of this method of precipitation, the great bulk of the milling waters used are continuously regenerated, and can be used over again and again indefinitely, and if the above specified quantity of lime has not been exceeded, the regenerated supply is absolutely soft, and requires no further treatment to render it fit for immediate re-use."

Experiments in wooden vats showed that downward percolation is sufficiently rapid through a layer of 12 to 18 in. of such a precipitate. With a continuous gentle upward flow of

leaching liquor under a head of 2 to 3 ft., all tendency to packing in a depth of 18 in. seems to be avoided. The specific gravity of the precipitated slimes is not very greatly more than that of the suspending medium (the water itself), and the upward leaching current tends to sustain and buoy up the particles of slime, and thus to prevent them from packing by their own weight. Should channels form, and the leaching be unequal, occasional slight agitation suffices to remedy it.

The small quantity of calcium oleate produced throughout the mass of the slimes appears in no way to interfere with the solvent action of the cyanide solution upon their gold contents. Dilute cyanide solution will remove from such a precipitate by direct leaching 90 % of the assay value of the original slimes. From the fact of the highly divided nature of the slimes, their gold contents are exceptionally free, and very susceptible to solvent action, so that much weaker liquors can be used than for ordinary tailings.

As to the quantity of lime necessary to precipitate the slimes from an effluent which has been treated at an earlier stage with soap, all that is required to effect complete deposition in one uniform leachable precipitate, is to mix with the waters from the classifier just sufficient milk of lime to decompose all the soap present therein with the formation of calcium oleate; 1 cwt. of soap (60 % fatty acids) requires just 10 lb. of lime (reckoning the latter at 80 %), the cost of which in England is, roughly, about 1*d.*

Several ways of proceeding are exemplified by Sulman:—

(1) Addition of soap solution to the whole bulk of ore and soft milling water in the battery stamps; classification-separation of the tailings; precipitation of the effluent slimes by adding milk of lime; and treatment of the precipitate (after deposition, and removal of clear effluent), by digestion with ·05 % and ·025 % solutions. Separation of the gold-bearing liquor, and subsequent washings with still weaker liquors, or water, are effected by decantation.

(2) No previous treatment of the milling waters is undertaken, nor is soap added to the pulp issuing from the battery;

but after separation of the sands in the ordinary way, the classifier effluent is collected in a tank, and a small addition of soda ash or caustic soda is there made, in order to render insoluble any dissolved lime or iron salts. A solution of soap is mixed with the liquor in the proportion of 2 or 3 lb. per ton of dry slimes present, and the precipitation of the whole is then effected by a small quantity of milk of lime. The subsequent treatment of the precipitate by dilute cyanide and decantation is the same as in the first method.

(3) The slimes are allowed to deposit from the effluent liquor by gravitation in the usual way; they are then treated with dilute cyanide direct, or they may be subjected if necessary to a previous soda treatment. As decantation of the gold-bearing liquor from the exhausted slimes would be impossible in the ordinary way, the requisite small quantity of soap solution is in this case added when the extraction of the gold is complete, and the final precipitation of the suspended (but depleted) slimes is effected by the admixture of milk of lime, the clear liquor being decanted off in a few minutes, as usual, and run through the gold-recovery plant.

In connection with the last-named method, it may be pointed out that the action of the cyanide takes place upon the slimes before the formation of the oleate of lime coagulum. There is no great advantage in this, as both soda and lime soaps are absolutely without chemical action, either upon potassium cyanide or potassic-auro-cyanide, and it has been proved that the presence of the insoluble lime coagulum offers no mechanical hindrance to the solution of gold in potassic cyanide. As, on the other hand, only the coarsest slimes are obtained by mere gravitation subsidence, a considerable bulk of the finer ore particles, undoubtedly carrying gold, still escape in the final turbid effluent.

In the decantation method, where the soap solution is applied only to the slime effluents, the saving of any float material which may have been produced upon the pulp surfaces during crushing is also effected. In this case, although such films have escaped amalgamation treatment, they are retained by the coagulum produced by the combined action

of soap and lime, and being thus thrown down with the general bulk of precipitate, are rendered available for solution by cyanide.

In all cases, the expenditure of 3 to 5 lb. of soap per ton of dry slimes is said to be "amply sufficient to effect their rapid precipitation by any of the above methods; taking the cost of soap at 2*d.* per lb. and reckoning slime production at 30 % of the original ore, this gives a maximum cost of—

| | | |
|---|---|---|
| Soda ash and lime | . . . . . . | 2*d.* |
| Labour | . . . . . . . . . | 1*d.* |
| Soap, 4 lb. @ 2*d.* | . . . . . . | 8*d.* |
| | Total . . . . | 11*d.* |

30 % of which is 3*d.* = cost of slime precipitation per ton of original ore."

A most convincing array of facts is put forward by the inventor in support of his coagulation process, yet, though now some 7 years old, it does not seem to have come into practice. In fact it has been asserted by W. Bettel that the consumption of soap on Transvaal slimes is 14 lb. per ton instead of 4 lb., and that the price is 4*d.* per lb. instead of 2*d.*, making the cost of the process "4*s.* 8*d.* per ton for soap alone," the results achieved being no better than by the Williams process (see p. 579), using 3*d.* worth of lime per ton.

*Decantation.*—Of the impossibilities of decantation with really obstinate slimes, there can be no doubt whatever.

Some experiments by A. James,* in which fine slimes were agitated with very dilute cyanide solution and then allowed to settle, showed that after 50 hours only 10 % of the liquor was sufficiently clear to be decanted; so that, supposing 1 oz. of gold had been dissolved from the slimes, it would require 22 agitations and decantations, lasting for nearly 46 days, before 90 % of the gold actually in solution could be separated from the slimes.

Again, in the author's experience at Lucknow, New South

* Trans. Inst. M. & M., iii. 94 (1895).

Wales, slimes after agitation with KCy liquor showed no disposition to even commence settling at the end of 4 days.

Nevertheless, in other cases, it is successful.

Estimating Solids.—In the decantation process as carried out in many Transvaal mills, the slime pulp flowing from the hydraulic classifiers automatically receives a regulated amount of lime, varying from $5\frac{1}{2}$ to 20 lb. to the ton of dry slimes, or $\frac{1}{2}$ to 1 lb. per ton of pulp.

The proportion of dry slimes present in the pulp is found by filling a bottle with the mixture and weighing the bottle, when a simple equation gives the amount of dry slimes; thus, if

$a$ = weight of bottle full of water,
$b$ = weight of bottle full of slimes pulp,
$c$ = dry weight of slimes in bottle,
$d$ = specific gravity of dry slimes,

then, on placing a known weight of dry slimes in a bottle previously filled with water, and weighing, $d$ is arrived at, thus

$$\frac{c}{a + c - b} = d, \text{ and } c = (a + c - b)\ d.$$

So that, if the bottle full of water weighs 1500 gr. ($a$) and the same bottle filled with slime pulp weighs 1650 gr. ($b$), and there have been added 250 gr. dry slimes ($c$), the specific gravity ($d$) will be

$$\frac{250}{1500 + 250 - 1650} \text{ or } \frac{250}{100} = (d)\ 2\cdot 5 \text{ sp. gr.};$$

and

$$(1500 + 250 - 1650)\ 2\cdot 5 = (c)\ 250 \text{ gr. dry slimes.}$$

The weight of water in bottle divided into 100 ($c$) gives the percental weight of dry slimes in the pulp; and by using a bottle holding a known weight of water, such as 10,000 gr., ($c$) would only need to be divided by 100 to indicate percentage. It is easy to prepare a table showing percentages at a glance. The water should be boiled after addition of dry slimes to dispel entangled air, and then cooled before weighing.

Lime.—The lime-precipitation and decantation process is often called the "Williams" method, from its having been first introduced by J. R. Williams at the Crown Reef. A diagram illustrating the arrangement of the plant is shown in Fig. 163, but many modifications in detail have been made. The inflow of slime pulp, freed from sands by previous classification, and mixed with the necessary amount of lime water delivered from an overhead tank is by way of a launder *a*, delivering to a series of conical vats *b*, provided with capacious outlets at the apex of the cone. Here settlement takes place, the supernatant water flowing away by launder *c* to catchment dams for re-use, while the sediment is discharged into the pipe system *d*, furnished with a steam-pump *e*, by which it is forced up into the agitation vat *f*. Here further settlement occurs, and more clear water is drawn off, by the series of taps *g* as shown, in some installations, but preferably by means of decanting siphons. When thoroughly settled, there will be about 20 t. of wet slime in a 100-t. tank, the 80 t. representing water drawn off. Weak KCy liquor is next run in, and agitation is produced by the centrifugal pump *h* for 10 to 30 minutes usually. Sedimentation is again permitted to take place, and the gold-cyanide liquor is similarly withdrawn into a vat *i*, for conveyance to the precipitators.

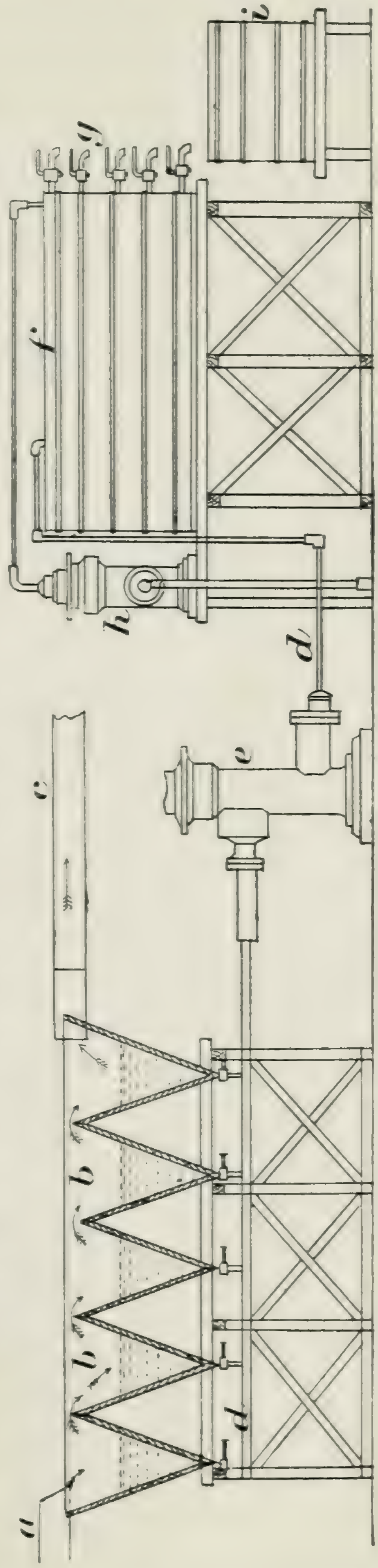



FIG. 163.—WILLIAMS LIME PROCESS.

In the most modern plants, the incoming slime pulp, carrying $2\frac{1}{2}$ to 5 % dry solids, is raised to $7\frac{1}{2}$ to 12 % in the conical vats, which have sub-level reception for pulp and rim-launder overflow for water, each vat being calculated to accommodate a definite quantity, such as 6, 12, or 24 hours' output. Subsidence occupies a period of $2\frac{1}{2}$ to 12 hours, or even longer, depending on the size of the vat and the nature of the material treated. A vat holds about 1 t. of slimes to 4 or 5 of water, and as the level of the settled slimes descends, the supernatant water passes off through the swivel-pipe or siphon, which is adjusted to continuously decant liquid until there remains only a slimey mass containing about 50 % moisture.

Weak KCy liquor (·01 to ·02 %) is pumped into the vats at great pressure, through a hose with a nozzle, to the amount of about 4 t. of solution for every ton of slimes, and this washes them through the opening at the bottom of the vat into the intake of a centrifugal pump, which is preferred to agitation by mechanical stirrers, as it ensures every particle of slimes being broken up and thoroughly mixed with the cyanide solution. The contents are sluiced out into the pump, and effective aëration is obtained, particularly if an air valve is introduced into the suction of the pump.* The agitation vat may be fitted with an arrangement for keeping the slimes in constant agitation for as many hours as are found to be necessary for the effective dissolution of the gold. After the slimes are settled, and the gold-cyanide liquor is decanted off for precipitation, the settled mass is sluiced into another vat, with weaker KCy solution, to go through a further decantation; and if the slimes are rich, this washing may be repeated.

Settled in deep tanks, slimes may contain less than 40 % of water. They are finally disposed of by sluicing, by washing into the intake of a pump, or by agitating with sufficient water to make a liquid pulp which will flow away to the residues dam.

The KCy solutions are used in advancing series, the final washing water becoming the weak solution of the next charge,

* An ill-advised proceeding: see p. 535.

and the weak solution of a previous charge (made up in strength) becoming the strong solution of the next charge. Thus the necessity for precipitating gold from a large volume of extremely poor liquors is obviated, and the quantity to be dealt with is kept within bounds.

Among the best results yet attained by this process are those at the Crown Deep, where, in 11,280 t. of slimes, assaying 3·157 dwt., an extraction of 87·49 % was got with 21,842 t. of solution, at a working cost of 2*s.* 9*d.* per ton.

Elsewhere, the cost incurred is thus given :—cyanide, 4*d.* ; lime, 3*d.* ; labour, 9*d.* ; wear and tear, 1*s.* 6*d.* ; fuel, stores, power, etc., 1*s.* 2*d.* ; total, 4*s.* per ton.

At another plant, actual extraction was 89 %. The average extraction is, however, considerably less, due to the low value of the slimes.

Water consumed by the method is stated to amount to about 50 % of the slimes treated ; but it is safer to estimate that a ton of water is thrown away for every ton of slimes, amounting to 2 or 3 times as much as is required by filter-press treatment.

Experimental treatment of W. Australian slimes by Alfred James showed no difficulty in getting a sediment containing 60 to 50 % dry slimes, and 91 % recovery was obtained in one instance by three cycles of agitation and washing, even with water carrying 6·2 % salt, ·45 % magnesium chloride, and ·73 % sulphate of lime.

An imitation of the Williams process has been introduced at Croydon, Queensland, by T. D. Kellaway,* as follows :— The slimes are trucked down to the works, and there the required quantity of quicklime to neutralise the acid contained in them is added as the trucks are emptied into a hopper discharging into a disintegrator. The lime required per ton of dry slime varies from 20 to 60 lb. After passing through the disintegrator, the powdered mass is raised by a bucket elevator to a top hopper, which discharges into side-tip trucks running over the tops of the vats. Over each vat is a 6-in. centrifugal pump, the suction pipe of which descends

* Min. Jl., Nov. 1899.

at an angle to within 2 ft. of the bottom of the vat, while the discharge pipe is continued down to the floor of the vat, where it is terminated by a right-angle bend on a swivel point, so that the flow can be directed across to any part of the vat. The vat being filled with solution of cyanide, the centrifugal pump is started, and in a few minutes the liquor is in violent agitation. Into this agitated liquor the powdered slime is dropped, truck by truck, until the charge, amounting to 10 t., is delivered, the agitation being maintained. A few minutes after the last truck is dropped in, the pump is stopped, and the mass is allowed to settle; this occupies some 20 minutes, the solution being left absolutely clear on top. A pipe, mounted with swivel joint, and connected with the pipe service leading to the extractor boxes, is then gradually lowered till the supernatant solution is drawn off. The vat is then filled with weak solution, the pump is once more set in motion, and the first process is repeated. After the second solution has been drawn off, clean water is added; and when the mass is worked into a slurry, a plug in the bottom of the vat is opened, and the thin slurry is run off in wooden launders, whence it is raised and delivered away by a large centrifugal pump. On 194 t. treated, assaying 8·44 dwt. gold and 2·55 oz. silver, giving a gross value of 40*s.* 11*d.* per ton, the average residue assay is 1·79 dwt. gold and 28½ dwt. silver, or a value of 9*s.* 10*d.* The cost is reckoned at 11*s.* 7*d.* per ton. Lime is obtained by burning local sea shells, and costs about 5*l.* per long ton.

**Agitation.**—That agitation greatly hastens the process of dissolution, as compared with percolation, is admitted on all hands. Experimenting on Woodstock (New Zealand) ore crushed wet to 90 mesh, assaying 10 dwt. gold and nearly 14 oz. silver per ton, McConnell * obtained the following results:—

With 1·2 % KCy solution:

Percolation: 70 hours; gold recovery 70 %, silver 23·4 %.
Agitation: 16 ,, ,, 90 %, ,, 83·8 %.

* Trans. Inst. M. & M., vii 31 (1899).

With ·6 % KCy solution :

| | | | |
|---|---|---|---|
| Percolation : | 70 hours ; gold recovery | 64 %, | silver 12·5 %. |
| Agitation : | 16 ,, ,, | 82 %, | ,, 66·5 %. |

The consumption of KCy was in each instance less by agitation than by percolation.

Further tests on the efficacy of duration of agitation resulted as follows :—

(*a*) Ore assaying ·55 oz. gold and 7·8 oz. silver per long ton.
KCy solution : 1·2 % reduced to 1·1 %.

| | | | |
|---|---|---|---|
| Tailings assayed after | battery : | ·21 oz. gold, | 5·84 oz. silver. |
| ,, ,, | 1 hour : | ·18 ,, | 5·57 ,, |
| ,, ,, | 4 hours | ·10 ,, | 4·68 ,, |
| ,, ,, | 7 ,, | ·05 ,, | 3·55 ,, |

Maximum extraction : 91 % gold, 54·5 % silver.

(*b*) Ore assaying ·42 oz. gold and 7·98 oz. silver.
KCy solution : 1·2 % reduced to 1·11 %.

| | | | |
|---|---|---|---|
| Tailings assayed after | battery : | ·24 oz. gold, | 6·39 oz. silver. |
| ,, ,, | 2 hours : | ·18 ,, | 6·05 ,, |
| ,, ,, | 3 ,, | ·16 ,, | 5·89 ,, |
| ,, ,, | 6 ,, | ·11 ,, | 4·92 ,, |
| ,, ,, | 8 ,, | ·08 ,, | 3·72 ,, |

Maximum extraction : 81 % gold, 53·4 % silver.

(*c*) Ore assaying ·7 oz. gold and 9·4 oz. silver.
KCy solution : 1·2 % reduced to 1·11 %.

| | | | |
|---|---|---|---|
| Tailings assayed after | battery : | ·15 oz. gold, | 7·45 oz. silver. |
| ,, ,, | 1 hour : | ·15 ,, | 6·50 ,, |
| ,, ,, | 4 hours | ·10 ,, | 7·20 ,, |
| ,, ,, | 7 ,, | ·08 ,, | 7·07 ,, |
| ,, ,, | 11 ,, | ·08 ,, | 6·47 ,, |

Maximum extraction : 88·6 % gold, 31·2 % silver.

(*d*) Ore assaying ·48 oz. gold and 7·75 oz. silver.
KCy solution : 1·2 % reduced to 1·08 %.

| | | | |
|---|---|---|---|
| Tailings assayed after | battery : | ·25 oz. gold, | 6·13 oz. silver. |
| ,, ,, | 2 hours : | ·16 ,, | 4·89 ,, |
| ,, ,, | 3 ,, | ·13 ,, | 5·20 ,, |
| ,, ,, | 5 ,, | ·09 ,, | 4·36 ,, |
| ,, ,, | 7 ,, | ·08 ,, | 3·27 ,, |

Maximum extraction : 83·4 % gold, 57·8 % silver.

(*e*) Ore assaying ·53 oz. gold and 8·43 oz. silver.
KCy solution : 1·2 % reduced to 1·1 %.

| | | | |
|---|---|---|---|
| Tailings assayed after | battery : | ·32 oz. gold, | 7·68 oz. silver. |
| ,, ,, | 1 hour : | ·20 ,, | 6·53 ,, |
| ,, ,, | 3 hours | ·16 ,, | 5·72 ,, |
| ,, ,, | 6 ,, | ·09 ,, | 5·44 ,, |
| ,, ,, | 15 ,, | ·05 ,, | 2·20 ,, |

Maximum extraction : 90·6 % gold, 73·9 % silver.

All the ores were treated in 2-t. lots, wet-crushed to 30 mesh with KCy liquor and agitated with a portion of the same; they contained not more than ·3 % pyrites, a little manganese, and a trace of copper. Much of the silver in these ores is present as sulphide (see p. 491), which in a measure explains the advantages of agitation. Percolation for 6 days did not extract over 65 % of the value, and caused just as much loss of cyanide. This indeed amounts (with liquors at 2 to 1) to 5 lb. per ton, and is probably also due to the silver sulphide and other cyanicides in the ore. As a rule, however, mixed tailings (including sands) do not require or warrant the extra cost of agitation methods.

Barrels.—Agitation in barrels with KCy solution, and subsequent leaching, was adopted in 1893 by the Golden Reward Co., Black Hills, in lieu of chlorination for such of their ore as carried much silver. It seemed to the author from conversation with the company's chemist, Mr. Bamberger, that this method was adopted chiefly because there was no accommodation in the mill for vat treatment, and partly perhaps because the barrel system of chlorination had been successful in his hands. On small parcels, at any rate, Bamberger got better results than an adjacent mill using vats; but the comparison was misleading, because the latter installation was crude and faulty in every respect, and could only be regarded as a total failure, so that superiority over it did not necessarily mean a great achievement. Whether, since the Golden Reward works were burned down, barrels have again been installed, the author has been unable to learn. Elsewhere, however, it would appear that barrel cyanidation has not survived first trials.

Mechanical stirrers.—The practice at the Silver Star mill, Montana,* is to shovel the slimes into an agitator 16 ft. diam. and 8 ft. deep, with a vertical shaft bearing 2 blades, driven at 18 rev. per min. The usual charge is 12 t. of very compact slime, increasing to 20 t. when less cohesive. The volume of solution used is 1 to 2½ t. per ton of ore, varying with the character of the slime and the proportion of solution which

* M. W. Alderson, En. & Min. Jl., Dec. 24, 1898.

can be decanted ; the strength of solution is such as to test 1 lb. KCy per ton during agitation. After 12 hours' agitation, the solids are allowed to settle, and the clear gold cyanide liquor is drawn off ; a second and a third wash is given, and the exhausted liquors are fortified for re-use.

The South German mill, Maldon, Victoria, has agitation vats 18 ft. diam. and 4 ft. deep, fitted with what is known as Deeble's patent agitator, though what there can be to "patent" in a tub carrying a vertical spindle with 4 arms is a puzzle. The vats are first filled, according to W. B. Gray,* with 2 ft. of sump solution, the agitator is set in motion, and 15 t. of slimes are charged in, bringing the level of the contents to within 3 in. of the top of the vat. Then 80 lb. of lime are added, partly as an alkaline agent, but chiefly to aid in subsequent deposition of the slimes. Next is introduced as much KCy as will bring the solution up to about ·15 %, and agitation is continued for at least 30 hours. The agitator is stopped and raised out of the contents of the vat, which settle during the following 8 to 16 hours. As settling proceeds, the clear supernatant liquor is drawn off by depressing the outlet gate, and run through a sand filter so that it is quite clear and bright on entering the precipitators. The gate is raised, the first wash is run on (weak sump liquor) and agitated for ½ hour ; then drawn off in similar fashion to the first liquor. A second sump solution and a final water wash are given, the volume of the last being so gauged as to keep the total solution in circulation at a constant figure. The exhausted residues are sluiced out. On visiting these works in 1899, the author noticed that the mechanism for raising the agitator consisted of windlasses rigged on a staging, and requiring 2 men for their operation, which compares very unfavourably with the counterpoise and chain arrangement adopted at Lucknow, needing only a boy, and capable of manipulation while the agitator is in motion. The issue gate also is more cumbersome and less automatic than the floating siphon or swivel pipe.

Pumps.—The employment of centrifugal pumps to effect

* Trans. Aust. Inst. Min. Engs., v. 138 (1898).

agitation has been already mentioned in connection with the Williams lime process, and the modification of this by Kellaway, which seems to have been awarded a Colonial patent, though why is not very apparent (pp. 579, 581). H. T. Durant advocates a centrifugal pump, with special air valve under control, for extra aëration. (See p. 535.)

Agitation by pumping is said to cost less than 4*d*. per ton. Nevertheless, specifications for a 60-t. (per diem) plant embrace two 40-h.p. boilers working at 100 lb. pressure, and a 40-h.p. steam engine.

Pneumatic.—The need of abundant air during the reactions between gold and KCy liquor has suggested the use of compressed air as at once a medium for agitation and aëration, and a so-called "pneumatic" process has for some time been before the American public. According to it, the "leaching vats are filled with crushed ore and flooded from the bottom with cyanide solution in the usual manner, using sufficient solution to make the mass of about the consistence of thick mortar; the valve regulating the amount of pressure is then opened, and a current of compressed air is gently turned into the pulp, being evenly distributed by means of a coil of perforated pipes. It rises bubbling upward through the whole mass of saturated ore, keeping every particle of the solution in motion and contact with both ore and air, and thus furnishing both the oxygen and agitation needed for rapid and thorough work in leaching. The coil of perforated pipe may be placed either on top of the filter cloth, or between the true and false bottoms, its position being determined by the character of the ore. For ores of a talcose or clayey nature, and showing a tendency to disintegrate and settle as mud or slimes on the false bottom, the compressed air may be turned on so strongly that the whole mass can be kept seething and boiling until settling is impossible. This process also permits of finer crushing than is usual in cyanide treatment, which means quicker and higher extraction, and enables lower grades of ore to be handled profitably, because the agitation produced by the compressed air prevents all packing and channelling, and the slimes, which, in ordinary methods inter-

fere with percolation, are forced by the compressed air to the surface, leaving only the coarser and heavier portions of the ore on the filter cloth." Apart from the very open legal question whether the use of compressed air in this manner can be laid exclusive claim to under patent rights, there are chemical, mechanical, and economical considerations which rob pneumatic agitation of that ideal character which it theoretically seems to possess. Firstly, there is no chemical demand for any such volume of oxygen as is thereby afforded—the supply under such circumstances is in enormous excess. Secondly, it is impossible to obtain anything approaching uniformity in the agitation. The classification produced by the air currents, as claimed by the advocates of the pneumatic process, and undeniably occurring, is in itself evidence of stratification rather than agitation, and it will be found that the solid contents of the vat will soon become assorted, the coarsest material lying at the bottom and being overlaid in turn by successively finer matter until the topmost portion is impalpable slime. Air channels are formed through the mass corresponding with the apertures in the supply pipes, and, unless the size of these apertures is carefully graduated, or a large excess of force is used, most of the air will escape at the first holes it reaches and the distribution will be partial. Finally, compressed air as a means of agitation is a most expensive way of applying the power used, even if it were otherwise satisfactory. Extended trials were made by the author on pneumatic agitation for treating the difficult tailings at Lucknow, New South Wales, in 1898, and the idea was abandoned for the reasons given above.

The proprietors of the pneumatic process claim that further benefits arise from using the compressed air in a hot state, gained by passing it through a small furnace between the compressor and the leaching tanks. When the vat has been filled with pulp, and the accompanying water has drained off to the uttermost, hot compressed air is admitted from below, and, "rising upward through the mass of tailings, will heat them and drive off the remaining moisture in the form of steam. When the mass of tailings is thoroughly dried, it

is then in excellent condition for the cyanide treatment, which is proceeded with as in the case of dry-crushed ores." It is asserted that "the expense of heating the air is but nominal: at the North Star mine, Grass Valley, California, all the pumps, hoisting engines, etc., are run by compressed air, brought several thousand feet from the compressor, and re-heated before use, at an expense of less than 1 cord of wood in 24 hours" (or about 1*s.* per hour). The comparison, however, is obviously misleading, and the blowing of hot compressed air through the contents of the vats is never likely to supplant, on economical or any other grounds, the use of a steam coil or jacket in those cases where heating is necessary or desirable.

**Filtration.**—While coagulation followed by decantation (p. 573) has proved an economic success for slimes which are not very rich, and where water is fairly plentiful, other conditions are encountered. For example in W. Australia, water is not abundant, and the talcose clayey nature of the ores of the oxidised zones renders it impossible to crush them either by a wet or a dry process without the production of a large percentage of rich slimes, in many cases carrying as much as 1 oz. gold per ton of ore. As an instance of the richness of the slimes from some of these West Australian ores, W. McNeill* mentions that in some of the dry-crushing mills the finest impalpable powder which floats in the air about the mill buildings and is finally deposited upon the top of the roof rafters, assays 1 to 2 oz. gold per ton; and the same feature has been noticed in most dry mills. Hence the separated slimes are not adapted to the S. African method, but are treated in filter-presses, either by (*a*) agitation with cyanide solution and delivery of the sludge to the press, finally washing the charge in the press; or (*b*) agitation with preliminary water or other washes if desirable, delivering the resultant sludge to the press, and effecting extraction with the charge in the press by passing through it the cyanide solution, finally washing the charge as before.

*Presses.*—Fig. 164 shows the general arrangement of the

* Trans. Inst. M. & M., vi. 247 (1898).

plant at Hannan's Brownhill. The slimes are introduced into the vats *a* and agitated by paddle with cyanide solution as long as may be found necessary to dissolve out the gold. The contents of the vat are then run into the receiver *b* and forced by means of compressed air at a pressure of 60 to 80 lb. per sq. in. into the filter-presses *c*, the filtrate or effluent from the presses being delivered through the pipe *d* into the vat *e*, from which it flows in a constant stream through the precipitation boxes to the sumps. The cakes having been washed in position, so as to remove any cyanide solution they may contain, the press is opened, and the cakes are allowed

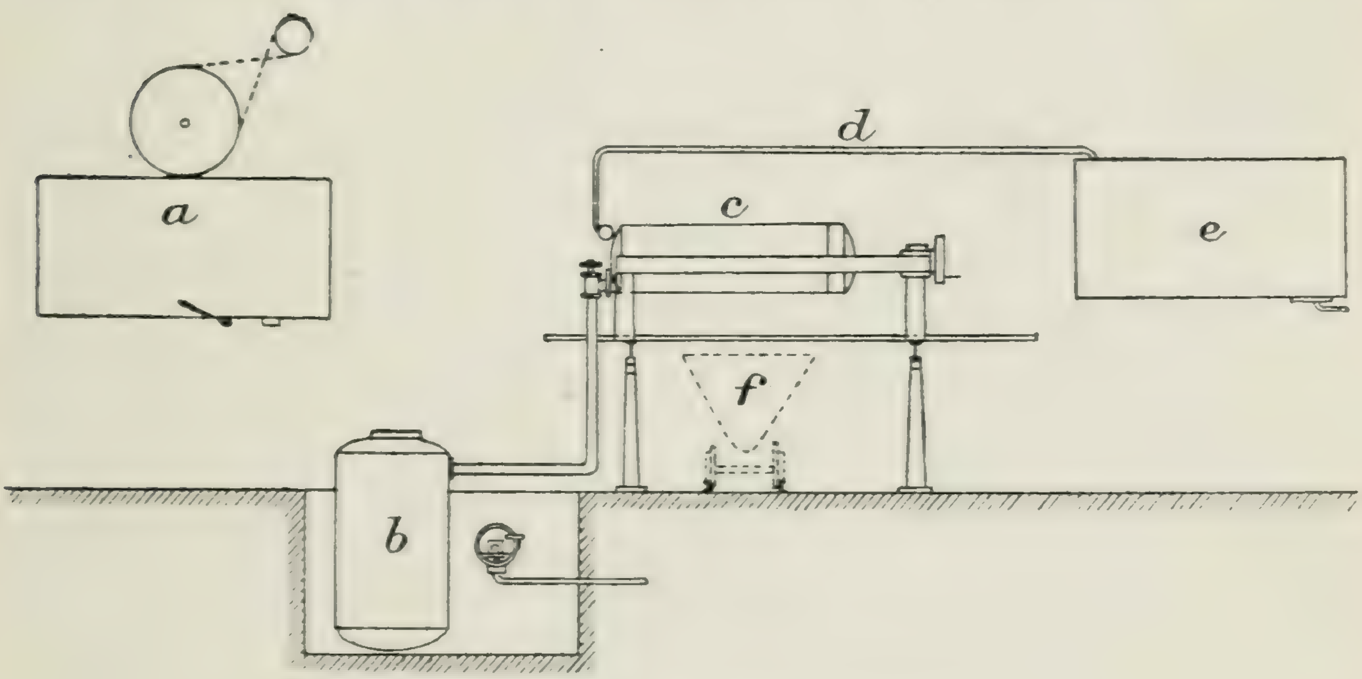



FIG. 164.—FILTER-PRESS PLANT.

to fall into the truck *f* for tramming to dump. The pneumatic forcing receiver or montejus *b* consists of a cylindrical steel vessel with dished ends. The pipe leading from it to the press reaches almost to the bottom of the receiver, thus ensuring, as soon as air pressure is turned on, the whole of the contents of the montejus being discharged into the press. Prior to discharging the receiver, the charge is stirred up by admitting a certain quantity of compressed air through a small pipe to the bottom of the receiver. The filter-presses are of the distance-frame flat-plate, and 4-eyed type, each press having 20 chambers forming cakes 28 in. sq. by 3 in. thick ; they are made by Johnson & Co., London.

McNeill found with the slimes of some W. Australian ores no difficulty in forming cakes of this thickness when

adopting dry crushing. The cakes were examined by A. C. Claudet and found to carry 20 % moisture and to weigh about 130 lb. per cub. ft., from which figures it will be seen that one press treats per charge about $1\frac{1}{4}$ t. of dry slimes.

The construction of the distance-frames and of the two side plates, which when placed together form a complete element of the press, is shown in Fig. 165. The 4 holes *a b c d* at the corners of the plates and distance-frames correspond when they are placed together in position, and form passages throughout the whole series of plates in the press. The side plates have cast upon their surfaces a series of fine grooves, so that, when the filter-cloth is placed over

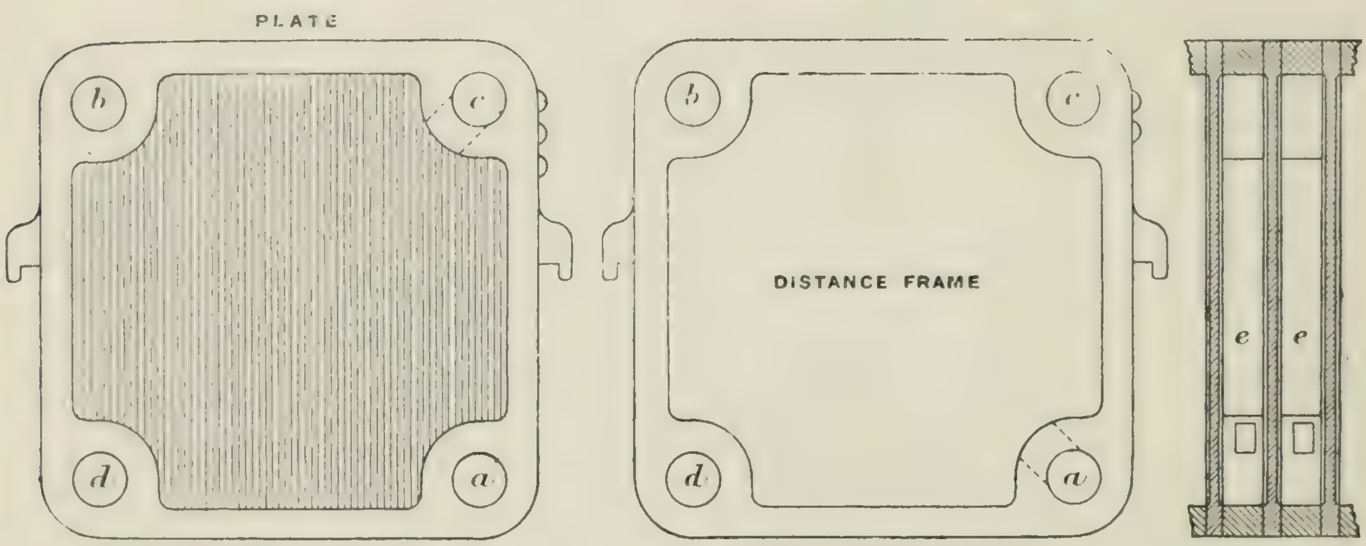



FIG. 165.—DETAILS OF FILTER-PRESS.

them, they form channels for the liquid, which is forced through the cloth, to pass along and out at the holes in the corners of the plates. But this type of construction is going out of favour, and is giving place to one in which the channels are external to the plates, the only connection necessary being a rubber ring between each two plates ; thus mounting and dismounting are greatly hastened.

The operation of filling and washing a charge in the press is as follows :—The sludge is forced by air pressure into the press from the receiver through the passage *a*, which extends through the plates and communicates with the interior of the distance-frame. The sludge continues to flow in until it completely fills the space *e* between the plates ; the liquid, forced out by pressure through the filter-cloths on either side of the plates, passes through the channels on the surfaces of the

plates to the openings *b c* and to the discharge pipe at the press head. The opening *d* would also serve for discharge of effluent liquid during filling, but, as it is connected up to the washing-pump, it is only used for passage of wash-water. After the cell has become compacted with material, and no more liquid issues, the sludge inlet *a* and valve on outside of press are shut. Water is then forced in at *d* and along the drainage channel on the surface of plate through the filter-cloth placed on this surface, and then through the cake, displacing the cyanide solution, and finally escaping through the filter-cloth on the opposite side, and out of the press through the opening *c*.

A press of this type admits of the cyanide solution contained in the cakes being effectively replaced by wash-water, and of any desired thickness of cake being secured by the width of distance-frame employed. The efficiency of the washing and displacement of the cyanide solution contained in the cakes by the wash-water, even on 3-in. cakes, is most complete, and uneven percolation does not take place.

While some slimes can be successfully dealt with in 3-in. cakes, others will barely tolerate 1½ in., so that actual experiment is necessary in each case to determine how much plant is required for a given output.

As regards time occupied and extraction obtained in filter-presses, experiments made by McNeill with 5-dwt. slimes gave tailings assaying 1 dwt. after 3 hours' agitation, and the following average time for the completion of the cycle of filtering, washing, emptying, and closing the press :—

| | |
|---|---|
| Filling press from receiver . . . . . | 15 min. |
| Washing cakes in press . . . . . . . | 19 ,, |
| Discharging and closing press . . . . | 16 ,, |
| | 50 ,, |

In the treatment of slimes from ore that has been wet-crushed, 6 to 8 t. of liquid are required by the decantation process for 1 t. of slimes. But by filter-press treatment, the amount and strength of cyanide solution can be regulated to suit the character of the ore under treatment, and usually 1½ times the weight of slimes is sufficient to reduce the pulp to such consistence that it can be readily forced into the presses.

Where water is not abundant, the very small amount of wash-water required is also of great importance.

It seems probable that the dry-crushed and classified material at Hannan's Brownhill contains much more fine sand than occurs in slimes from wet-crushed and classified ore, and that the leaching in the press is therefore much more rapid and satisfactory than can be generally counted upon. A ·3 % KCy solution is used, with agitation for 6 hours, the operations of filling the presses, leaching, washing, and emptying occupying 1½ to 2 hours.

The cost of filter-press treatment in W. Australia is said by H. C. Hoover to range from *7s.* 1½*d.* to *8s.* 11*d.* per ton. At the Lake View Consols mill, the slimes from wet battery pulp are treated in filter-presses for, according to E. S. Simpson,* an 85 % extraction on 8·9 dwt.; while the Co.'s report for 1898 gives 8313 t. yielding 2794 oz. gold (or an average of about 6·72 dwt. per ton recovery), at a mean cost of about *9s.* 6½*d.* per ton.

Improvements are being constantly suggested in the direction of cheapening the filter-press treatment, and according to Alfred James, of the Cyanide Plant Supply Co., which has furnished the most modern plants in W. Australia, the current practice most generally adopted is as follows :—

The slimes are agitated in mixing vats for a sufficient time to dissolve the gold, and aërations (and even dilute chlorine solutions) are employed to accelerate dissolution. The slimes are then run into a "montejus" and forced by compressed air through the presses ; 70 % of the accompanying water is immediately obtained quite clear and ready for return to the dam or reservoir. Water-washing is carried out on the cakes in the presses. The presses have capacities varying from 1½ to close on 10 t. per charge, the latter being the estimated capacity of the new presses for the sulphide plants. As the slimes from roasted ores are not so permeable as those from the porous oxidised ores, modifications may require to be made in those of the larger presses which are provided with wide distance-frames.

* Trans. Am. Inst. Min. Engs., xxviii. 808 (1898).

The tendency of recent years has been towards increasing the size of the filter-presses ; this has necessitated attention to the difficulties arising from handling such heavy masses of metal, as, for example, the head-pieces, which have required to be specially reinforced to stand the strain. An arrangement is now included in some of the best presses whereby the screw is eased of the heavy strain due to the weight of the head-piece, and a system of rollers or wheel bearings has been introduced to facilitate the moving of the plates and distance-frames. It has been found essential to have apparatus which fills the presses rapidly without intermission or shock, and pumps have therefore been superseded by montejus, to which an air jet has been added, to agitate the contents and prevent the settling of the coarser particles. The cycle of charging, pressing, washing, and discharging the contents of a press is completed in 1 to $1\frac{1}{4}$ hours, the discharging itself taking only about 20 minutes. The pressed cakes are sufficiently dry to be handled and trucked to the dump.

Instead of performing dissolution in the presses themselves, provision is now made for the dissolution of the gold before it enters the presses.

The extraction obtained by the press method is given at 82 to 92 %, and the most recent costs, quoted by James, are as follows :—

| | Average per ton. s. | d. |
|---|---|---|
| Superintendence and settling solutions | 0 | 6·832 |
| Discharging and filling presses | 2 | 10·353 |
| Compressed air | 0 | 9·663 |
| Turning zinc | 0 | 0·196 |
| General repairs | 0 | 3·037 |
| Assaying, retorting and melting | 0 | 2·956 |
| Zinc | 0 | 0·528 |
| Filter cloth | 0 | 3·121 |
| Cyanide of potassium | 0 | 10·748 |
| General stores and charges | 0 | 4·403 |
| Electric light maintenance | 0 | 3·488 |
| Total | 6 | 7·325 |

An installation of 12 $1\frac{1}{2}$-t. filter-presses has a capacity of about 120 t. of slimes per diem ; in practice, one press is found to treat 10 to 12 charges per day of 24 hours. It is suggested that an improvement and economy on present agitation

methods would be the use of centrifugal slimes pumps and conical tanks.

Double filter-pressing as described on p. 571, for removal of salts in solution when dealing with pyritic ore is not calculated to add more than 1*s.* per ton to the cost, as J. K. Wilson, of the Golden Horseshoe, says 2 men per shift and 2 presses will readily deal with the moisture of 100 t. daily.

The table below gives some particulars of the Johnson filter-press of "pyramid" pattern with recessed plates :—

JOHNSON FILTER-PRESSES.

| Size of Filter Plates. | Chambers. | Thickness of Cake. | Area of Filter Plates. | Approx. Weight. | Working Pressure. |
|---|---|---|---|---|---|
| in. sq. | No. | in. | sq. ft. | lb. | lb. per sq. in. |
| 12 | 6 | 1 | 12 | 532 | 150 |
| ,, | 12 | ,, | 24 | 700 | ,, |
| ,, | 24 | ,, | 48 | 1,036 | ,, |
| 19 | 6 | ,, | 30 | 1,456 | ,, |
| ,, | 12 | ,, | 60 | 1,900 | ,, |
| ,, | 18 | ,, | 90 | 2,354 | ,, |
| ,, | 24 | ,, | 120 | 2,800 | ,, |
| ,, | 36 | ,, | 180 | 3,696 | ,, |
| ,, | 48 | ,, | 240 | 4,598 | ,, |
| 24 | 12 | ,, | 96 | 3,024 | ,, |
| ,, | 18 | ,, | 144 | 3,808 | ,, |
| ,, | 24 | ,, | 192 | 4,480 | ,, |
| ,, | 36 | ,, | 288 | 6,048 | ,, |
| ,, | 48 | ,, | 432 | 7,616 | ,, |
| 27 | 12 | $1\frac{1}{4}$ | 122 | 3,696 | 100 |
| ,, | 24 | ,, | 244 | 5,376 | ,, |
| ,, | 36 | ,, | 366 | 7,056 | ,, |
| 30 | 24 | ,, | 300 | 5,712 | ,, |
| ,, | 36 | ,, | 450 | 8,624 | ,, |
| ,, | 48 | ,, | 600 | 11,536 | ,, |
| 36 | 24 | $1\frac{1}{2}$ | 432 | 12,656 | ,, |
| ,, | 36 | ,, | 648 | 16,688 | ,, |
| ,, | 48 | ,, | 864 | 20,720 | ,, |
| 48 | 24 | ,, | 800 | 32,000 | ,, |
| ,, | 36 | ,, | 1200 | 40,000 | ,, |
| ,, | 48 | ,, | 1600 | 50,000 | ,, |

Of the Dehne press, as used at the Lake View Consols, the annexed details may be of interest :—

DEHNE FILTER-PRESSES.

| Size of Plate. | Chambers. | Approx. Weight of Cake. | Price. | Thickness of Cake. |
|---|---|---|---|---|
| in. sq. | No. | lb. | *l.* | in. |
| 32 | 24 | 756 | 131 | $1\frac{1}{4}$ |
| ,, | 30 | 944 | 151 | ,, |
| ,, | 36 | 1134 | 171 | ,, |
| ,, | 42 | 1322 | 194 | ,, |
| ,, | 50 | 1574 | 222 | ,, |
| 40 | 24 | 1188 | 192 | ,, |
| ,, | 30 | 1484 | 226 | ,, |
| ,, | 36 | 1782 | 258 | ,, |
| ,, | 42 | 2080 | 288 | ,, |
| ,, | 50 | 2476 | 333 | ,, |

*Continuous Filters.*—The interrupted character of filter-press treatment, and the consequent limited capacity of the plant, with high cost for installation, has led to numerous efforts in the direction of securing continuous filtration. Two of these methods promise to attain a considerable degree of efficiency, and are therefore worthy of description, viz. Bonnevie's travelling filter-cloth, introduced in the United States, and Draper & Riley's filter drum in Australia.

The Bonnevie continuous suction filter, as shown in Fig. 166, consists of a substantial iron frame-work *a* supporting a suction box *b* and provided with carrying roll, guide roll *c*, and press roll *d*, necessary for operating and supporting an endless horizontal filter-cloth *e*, which moves over the 3-compartment open-top suction box *b* at the rate of 3 to 8 ft. per minute, passing over the right-hand drive roller *f*, then to the felt whipper *g*, tightener rolls *h*, guide roll *c*, left-hand drive roller *i*, under the press roller *d* and over the suction box again. The suction box is of sheet steel, and the open top is quite closely covered with light wooden rollers *k* to prevent sagging of the filter-cloth. On one side wall of each compart-

ment of the suction boxes are openings connected with an exhaust fan, which creates a vacuum of about 3 oz. under the entire surface of the filter-cloth covering the suction-box. This vacuum has been generally found sufficient to draw all

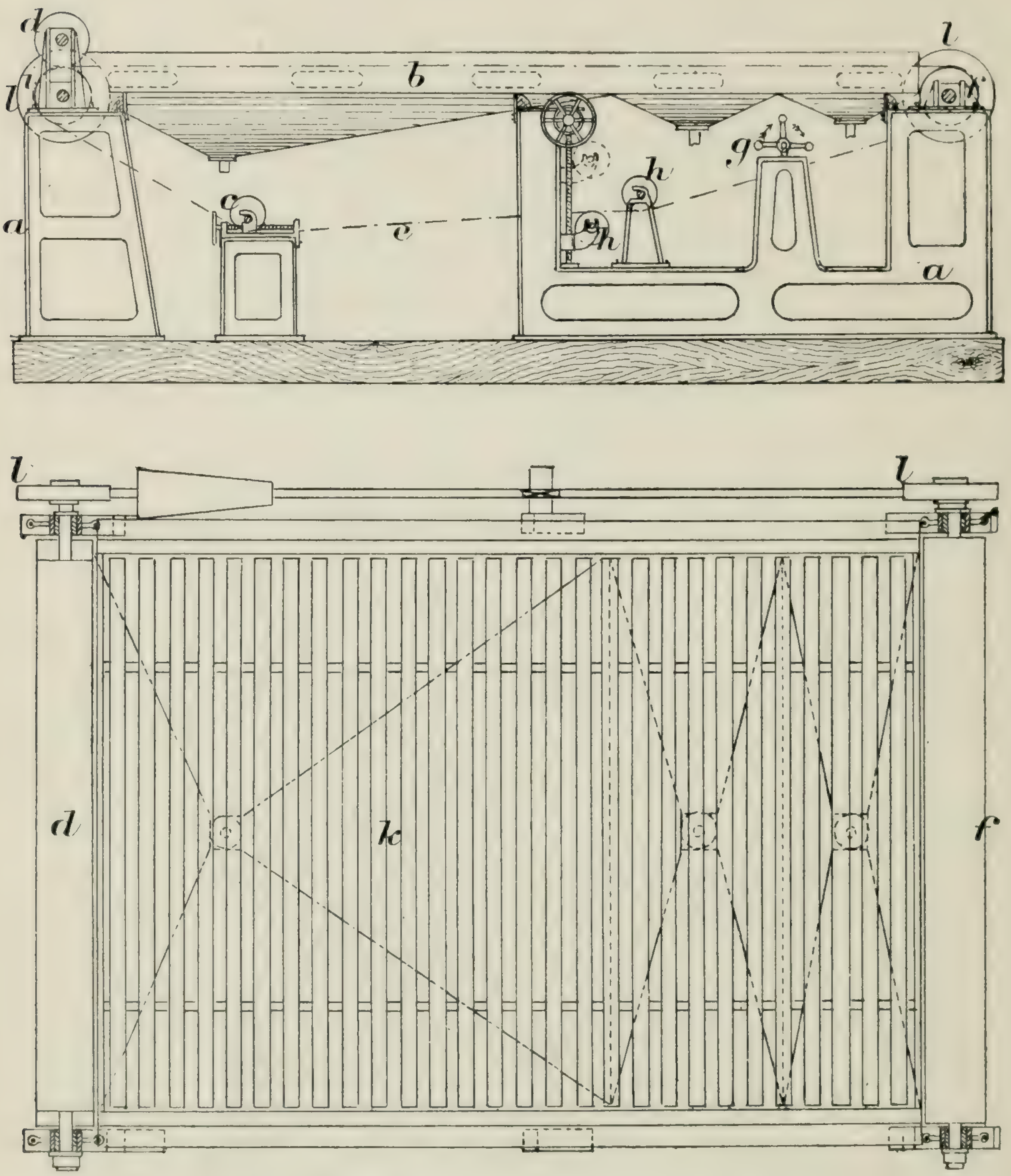



FIG. 166.—BONNEVIE CONTINUOUS FILTER.

moisture through the moving filter-cloth. The 2 end rollers are driven by worm gearing *l* (worm 1 in 40), from a countershaft (at 40 to 120 rev.), and cone pulleys are provided for easy adjustment of speed as desired. The guide roller is arranged to keep the cloth in a central position, the tightener-

roller for adjusting the filter-cloth tension, and the press-roller to remove the surplus moisture from the cloth. The felt whipper removes all tailings clinging to the surface of the filter-cloth after the final washing. The filter-cloth area is 8 × 12 ft., and it is calculated to deal with 50 t. per 24 hours.

The wet-crushed pulp is fed directly and continuously to the travelling filter, where it is deprived of surplus water. The relatively dry pulp is then washed from the filter-cloth by KCy solution of determined strength, and in necessary volume, and passes to agitation tanks ; or if required, it may be detached by clean water, or by alkaline wash, and then agitated. After agitation, it is again passed over the filter, and the liquor or wash, as the case may be, is here drawn off for further treatment. In this way, agitation and filtration are alternated with alkaline or water washes, and with strong or weak solutions, in a continuous sequence, till finished. Each compartment of the suction box has a separate outlet for liquids removed. The thickness of pulp on the filter-cloth does not exceed about ¼ in., so that permeation by the liquids should be easy. The feed once adjusted, and sufficient tanks supplied for agitation and for storage of various liquors, the operation is continuous.

As a rule, the operation of washing is as follows :—The solution in the pulp deposited on the filter-cloth, is drawn through it into the first compartment of the suction box. This takes place while the filter has moved 3 or 4 ft., leaving in the ore about 20 % of moisture containing values. To save these, 3 spray wash-waters are used, gradually displacing the solution and leaving only clear water in the tailings as they pass over the end of the filter. The first wash-water is drawn from the strong wash-water tank and is applied to the pulp as soon as the cyanide solution is drawn through into the first compartment of the suction box. This wash-water equals in quantity the amount of moisture left in the pulp, and merely displaces the original cyanide solution, leaving in the pulp only 20 % of strong wash-water. The liquid collected in the first compartment consists of practically all the cyanide solution containing values which was in the pulp when it was deposited on the

filter-cloth. This solution flows into the settling box and thence to the precipitation department. It is thence pumped back to the standard solution tanks, cyanide is added to raise it to the standard strength, and it is again fed with ore into the first agitation tank. The pulp on the filter-cloth now contains only the strong wash-water which was applied at the first compartment, together with a very small percentage of the original solution ; while it passes over the second compartment, the second wash of weak wash-water is applied, displacing this strong wash-water. The strong wash thus displaced flows from the suction box to a rotary pump, which elevates it to the strong wash-water tank, whence it is drawn again for the first wash-water. The pulp, as it passes to the third compartment of the suction box, contains only the weak solution wash-water, which is there displaced by the third wash, consisting of an equal amount of clear water. The weak wash-water thus displaced flows from the suction box to a rotary pump, and is elevated to the weak wash-water tank. The tailings passing over the end of the filter contain only about 20 % of clear water, all values having been removed by the three wash-waters. The volume of cyanide solution, strong wash-water and weak wash-water remains practically constant, the only loss being due to evaporation, slight leaks, etc. This is made up by a slight addition to the final clear water-wash over that required to displace the weak wash-water. This addition gradually advances through the weak wash-water, strong wash-water and finally into the cyanide solution, keeping the volume constant and carrying the increasing values of the weak and strong wash-water back into the cyanide solution. The solid residues returning under the suction box, now thoroughly washed, are subjected to a strong water spray that washes them into the sluicing box to be carried away, with provision for arresting any coarse gold. The filter-cloth, on its return under the suction box, is further subjected to a cleaning process, by means of the felt whipper and another water spray, and, reappearing on the surface, the press roll relieves it of most of its moisture, so that when it reaches the front of the apron it is clean, nearly dry, and ready to receive a new charge

of ore and cyanide solution. The settling box does away with the gold solution tank used in percolation as a regulator of

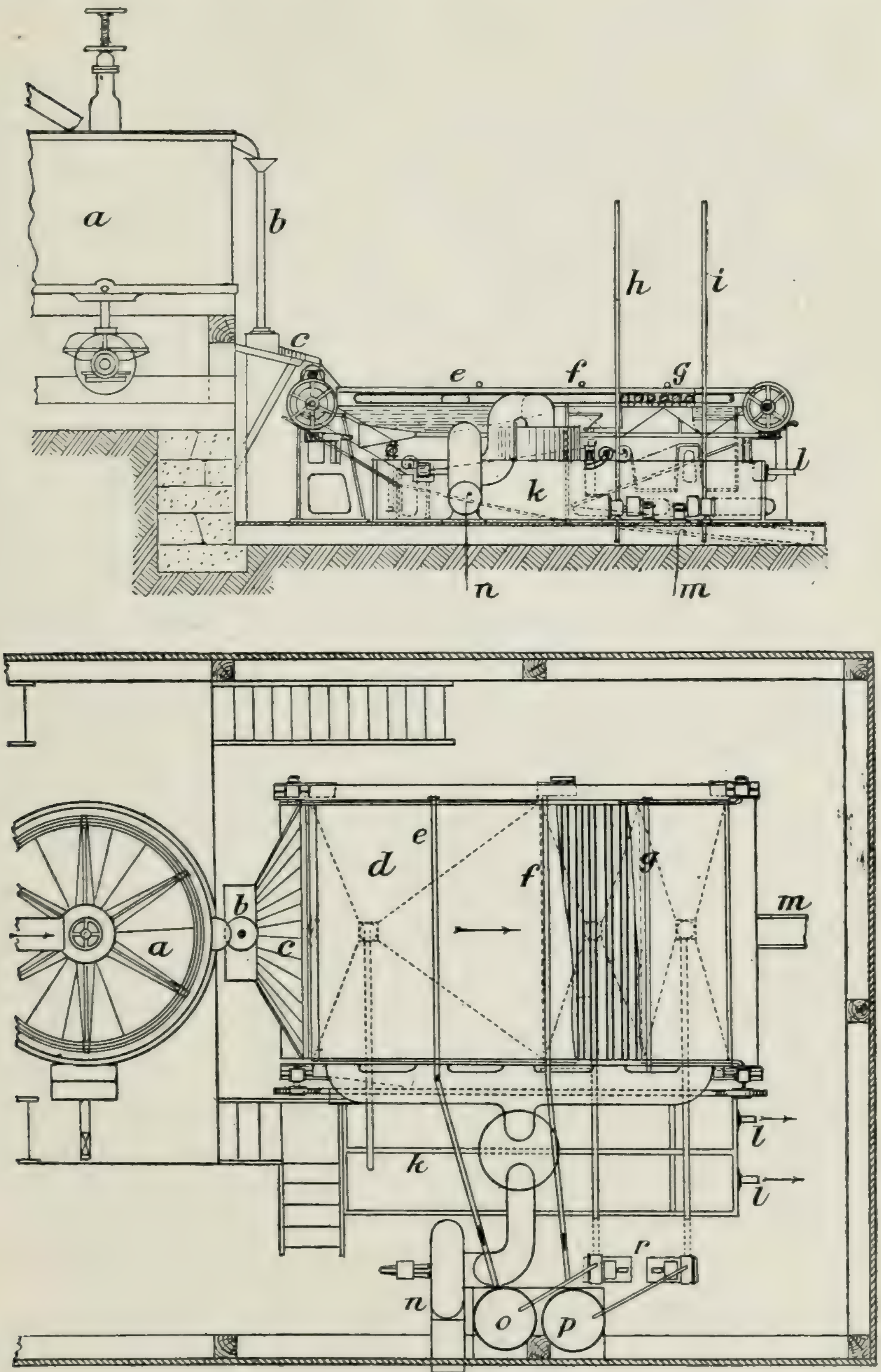



FIG. 167.—BONNEVIE PLANT.

the flow into the precipitation department, as the flow of liquor is always the same. The power necessary to run a 50-t. plant

is said to be about 20 h.p., the tanks requiring each 3, the filter 1, and the pumps and fan about 7 h.p.

The complete installation is shown in Fig. 167 :—*a*, last of series of agitating tanks ; *b*, feed to distributing apron *c* ; *d*, travelling filter-cloth ; *e*, perforated pipe bringing strong KCy liquor ; *f*, ditto for weak liquor ; *g*, ditto for water ; *h*, strong liquor return ; *i*, weak liquor return ; *k*, 2-compartment settling box ; *l*, pipes to precipitators ; *m*, residues sluice ; *n*, exhaust fan ; *o*, strong liquor tank ; *p*, weak liquor tank ; *r*, rotary pumps.

In Draper & Riley's machine, the travelling filter-cloth is stretched over a slowly rotating drum. The first example was made with a wooden drum, but this soon warped and gave trouble, and it has been replaced by sheet steel. To the interior of the drum, suction is applied, and the various liquors and washes are thus drawn through the coating of pulp lying on the filter. The exhausted residues are sufficiently cohesive, through drying by air, to adhere to the face of the belt till it reaches a scraper or brush, which detaches them into a truck for removal. Trial runs on several tons of Lucknow slimes assaying about 10 dwt., which are extremely difficult of treatment owing to their impervious character, indicate every probability of success. With a single liquor, all the gold is got into solution, and with washes of weak liquor and water the recovery has been 60 to 70 %. There is no doubt, in the author's mind, after many months' experiment on these refractory slimes, that by providing greater tank accommodation, so as to admit of agitation with the wash-waters, and pumping the agitated mass back to the rotary filter for separation purposes only, a most complete extraction will be got at a cost much below that of filter pressing.

**Precipitation.**—The gold carried away from the leaching vats in a state of solution is recovered by precipitation from the auro-cyanide liquor. Various substances may be used as precipitants. That principally in favour is metallic zinc, which may be in shavings or in a finely pulverulent condition, and may be lead-free, normally contaminated with lead, or de-

liberately plumbified. Electrolytic methods, so called, but not more so than the ordinary zinc process, are the Siemens-Halske (using a lead cathode) and the Andreoli (using an iron cathode). Wood charcoal is much employed in Australia. Aluminium and stannous chloride have been suggested and experimented with ; several forms of cyano-amalgamation have been practised in different places ; and copper sulphide has been proposed by Prof. De Wilde.

The concentration of gold liquors by repeated use before precipitation, originated by J. R. Williams at the Crown Reef, is now general, permitting a material reduction in the extractor plant and effecting economy in working. This concentration might be carried to the point when the liquor would almost equal the gold-contents of the material to be leached ; but in practice this is not desirable, and solutions are seldom used more than twice without precipitation.

*Clarification.*—Whatever the method adopted, the first consideration is to deliver to it the gold-cyanide solution in a clear condition, so that suspended matter may not interfere with the reaction. It is a singular fact that such an obvious desideratum as clear bright liquor, and one so easily attained, in many instances is quite ignored. Even with the utmost care in drawing off liquors, particularly from slimes, it is difficult to ensure absolute freedom from matter in suspension ; and in case of a little carelessness or mishap, the solution may come away quite cloudy, if not actually turbid. This may be very simply prevented by passing the liquor through a sand filter, either one large filter-tank of a capacity to accommodate the liquor from all the leaching tanks, or a separate small filter-tank for each. At Lucknow, the author adopted an arrangement which proved most effective, and is as follows. To a small square tank (in this case, half a 400-gal. iron tank used in shipping "soft goods" to the Australian colonies) was fitted a wooden frame, supported about 6 in. clear of the bottom, and carrying a stout woven wire screen, covered with burlap, and loaded with a few inches of quite clean sharp sand. The filter-cloth and its sand can easily be removed and washed when choked, and as easily replaced. A steam ejector hastens

the filtration if necessary. There is no difficulty in making this filter serve the purpose of controlling the rate of flow to the precipitators, on the same principle as the settling tank already described (p. 549).

*Zinc.*—It being desirable to expose a maximum area of zinc surface to the solutions, the metal is commonly employed in the form of very fine shavings or turnings. These may be purchased ready cut, but at most large mills the practice is to cut them as required, using V.M. brand, or one containing not more than 1½ % lead. Discs cut from No. 10 to 14 gauge sheet zinc and measuring about 12 in. diam. are now manufactured expressly for the purpose.

A convenient lathe is shown in Fig. 168. A number of

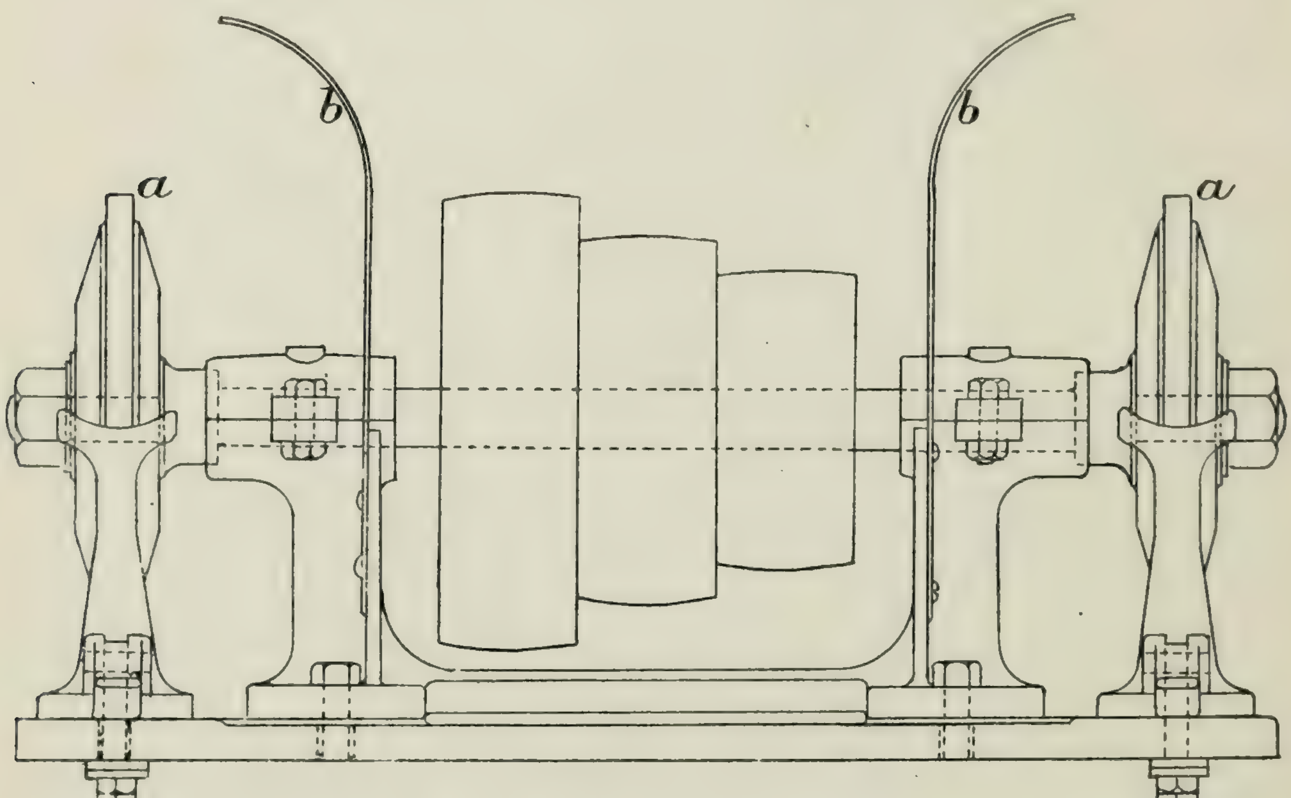



FIG. 168.—ZINC LATHE.

zinc discs (about 20) are clamped together on the mandrel by means of flanges, as at *a*, always using flanges of a size that will allow not more than 1 in. of zinc to project beyond the edges of the same, otherwise the discs will separate and cause more or less trouble when turning them. The lathe should be run so as to give the zinc a peripheral speed of about 1800 ft. per min. The tools used in turning are usually half-round files, about 12 in. long, ground down to a sharp cutting edge, or ordinary carpenters' mortising chisels. The zinc may be

turned down to about 3 in. diam. before the discs are discarded. Guards *b* placed at each end of the lathe prevent the shavings from flying out and injuring the belt, etc.

An automatic lathe is sometimes used, whereby nearly all the manual labour otherwise required is dispensed with. Thus at the Standard mill, Bodie, California, according to Bosqui,* the zinc is cut on a compound-geared machine lathe, provided with a wooden mandrel 3 ft. long and 9 in. diam., around which are tightly wound 5 sheets of commercial zinc, No. 9 gauge, 3 ft. wide and 7 ft. long, the edge of each sheet being soldered to the layer underneath. The side-cutting tool of steel is fixed to the carriage and adjusted to the edge of the core of zinc sheets. The mandrel is geared to 84 rev. per minute, or 242 rev. to 1 of the carriage-screw. Thus, in 15 minutes the carriage moves 1 in. and the mandrel makes 1260 rev. The shavings cut are 1/1260 in. thick and about 1/32 in. wide; 5 sheets of dimensions given weigh about 65 lb., and to cut this into shavings requires about 9 hours. Such a lathe, working automatically 4½ hours per day, will cut enough shavings to supply a 75-ton plant, operating under average conditions. As the zinc shavings run off from the lathe they are caught between a series of small iron rolls, operated from the mandrel, by which they are guided into a box under the lathe provided for the purpose; if allowed to wind around the mandrel, they must be periodically pulled off in bunches, which have to be disentangled before the material can be used for precipitation, consuming much of a man's time. Shavings cut as indicated present an area of about 1630 sq. ft. of surface per lb. of zinc cut up.

Usually the shavings are exposed in long boxes or troughs, so arranged that the gold-cyanide solution is forced by baffle plates to pass through a considerable quantity of zinc. The box is often made with 10 compartments, each of such dimensions that the amount of solution to be treated may flow up through the zinc easily and steadily, without eddies and without disturbing the precipitated gold. In practice, each compartment is designed to hold about 1 cub. in. of zinc for

* 'Cyanide Process,' pp. 99–101.

every ton per month treated. Thus a 2000-t. per month plant would have extractors or precipitators with compartments of such size as to contain about 2000-cub. in. of prepared zinc and this zinc space would be 15 in. wide, 15 in. deep, and 9 in. long (in the direction of the length of the extractor). The baffle boards are so arranged that the solutions always flow through the zinc in an upward direction. This ensures greater regularity of flow, assists in disengagement of the hydrogen gas given off, and increases precipitation of gold in the lower part of the extractor.

The precipitation box is built with a fall of 1 in 20 in the direction of its length. Though there are 10 zinc compartments, usually 6 only are filled with zinc. The first is used as a settler, when a special settling tank is not provided, to allow of the deposition of any matter coming away from the percolators with the solutions. The next 6 compartments are filled, and the 3 remaining act as settlers for any fine gold precipitate accidentally carried over from the last zinc compartment; or, when solutions are made up to their normal strength in the sumps, lumps of cyanide may be placed in the last compartment of the "strong" extractor box, which is thus converted into a dissolver. After passing through extractors, the solutions are conducted into their respective sumps.

Each compartment is fitted with cleats holding a tray of iron wire gauze of about $\frac{1}{8}$-in. mesh, on which the zinc rests, and through which the gold precipitate passes into the space provided for it below. At the lowest part of this space is fitted a plug, by the withdrawal of which the gold mud flows out of the extractor into the iron clean-up launder, the lid of which is fastened with good locks; and the top of the extractor is covered with a locked lid.

Such a box is shown in Fig. 169: *a*, inlet for solutions; *b*, baffles; *c*, wire trays; *d*, gold-mud plugs; *e*, clean-up launder; *f*, exit of solutions.

Unprotected iron is to be most carefully avoided in the extractors, especially where it comes in contact with the zinc, as the two metals form a strong galvanic couple, the effect of which is to send the zinc into solution and to gold-plate the

iron. To avoid the losses caused by this, porcelain trays have been suggested instead of iron wire gauze ; but they are too fragile and clumsy. Protective inert varnish may, however, be applied whilst the iron trays are hot.

An extractor as above for a 2000-t. plant would be about 12 ft. long, 18 in. wide, and 2 ft. 6 in. high (outside measurements). The sides, ends, and bottoms are made of clear yellow pine boards $1\frac{1}{2}$ in. thick ; the baffle-boards are $\frac{1}{2}$ in., except the centre and two intermediate ones, which may be of $1\frac{1}{2}$ in. wood, to give greater stability to the sides. All baffles are

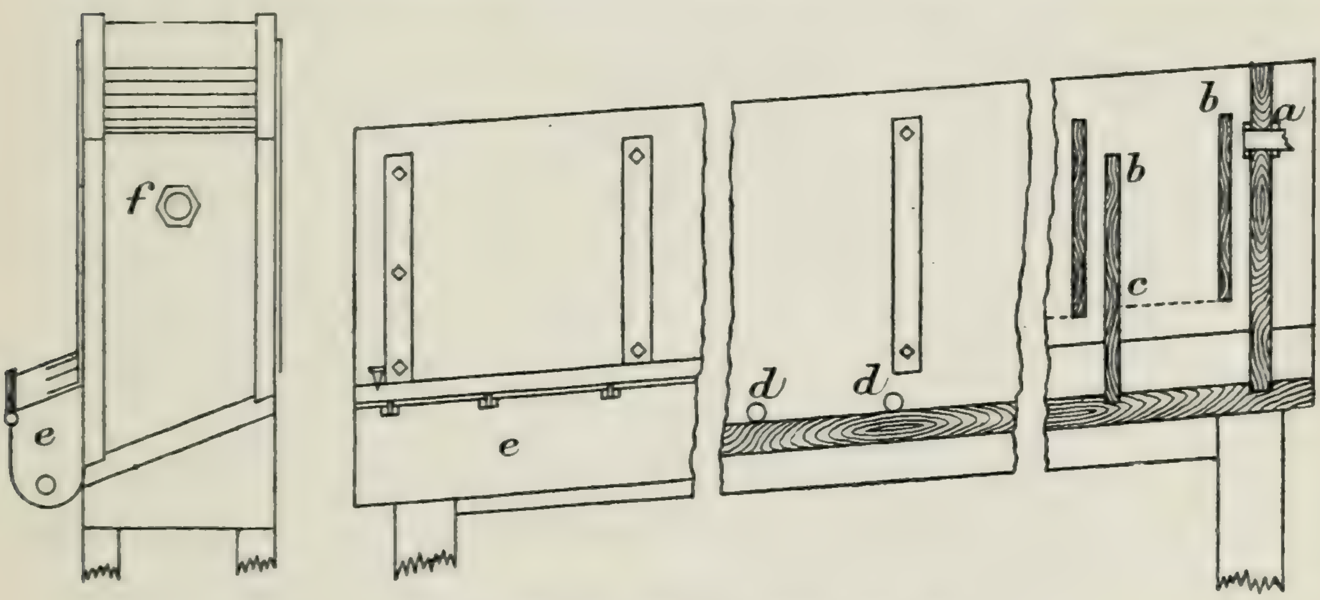



FIG. 169.—PRECIPITATION BOX.

gained $\frac{1}{4}$ in. into the sides, where they are held firm by screws, which also bind the sides ; the ends are kept together by bolts.

Not less than 2 boxes are necessary in each plant, one for strong and one for weak solutions ; but later practice is to employ 3, the additional one being for very dilute washings or surplus solutions. No solutions should leave the works—sluiced out residues excepted—without first passing through an extractor.

The consumption of zinc is about 4 oz. per ton of ore treated, though some works are claimed to use only 2 oz. A fresh supply is usually added to the boxes each 24 hours to replace current consumption.

The quantity of solution flowing through each box must be properly regulated. The only satisfactory method of knowing if proper precipitation is taking place is by having

the solutions regularly assayed. It is essential for the man in charge to ascertain the strength of solution at different stages of leaching, that he may know into which precipitation box it is to be passed ; and then he must test the strength of solutions leaving the precipitation boxes, in order to let them flow into their respective storage tanks.

The quantity of gold in solution increases and decreases in much the same way and same time as the cyanide in solution increases or decreases. For complete precipitation, an excess of cyanide in the solution seems to be essential.

According to W. Bettel, precipitation is influenced by the following conditions:—

(*a*) Different strengths of solvents.

(*b*) Impurities in solutions (which may be derived from ores).

(*c*) Rate of flow of solution through extractors, i.e. past a given area of zinc.

(*d*) Amount of gold in solution to be precipitated.

(*e*) Alkalinity, acidity or neutrality of liquors, affecting both rate and percentage of precipitation.

(*f*) Presence of such inert substances as carbonate of lime, clay, and oxide of zinc ; also coating of zinc.

(*g*) Other physical conditions of zinc affecting precipitation of metals from cyanide solution.

Complete precipitation cannot be attained with zinc shavings, but this is not a serious matter, if the solutions are never run to waste, as it simply means that a certain proportion of the gold is in perpetual circulation in the liquors. Still it is claimed that, on the average of Transvaal practice, extractor liquors seldom contain more than 6 gr. gold per ton ; and on other fields, where the solutions are run to waste, final effluents rarely exceed 2 gr. per ton, with cyanide present varying from traces up to ·07 %.

Some authorities consider that in zinc precipitation there is as much galvanic action as in so-called electrical processes, the iron of the wire grating forming a galvanic couple with the zinc, and beginning the action. Then, on gold precipitating on the zinc, a gold-zinc couple is formed.

In the ordinary conduct of operations, the solutions flowing from the leaching tanks, whether percolators or agitators, are tested both for gold and for cyanide, and are conducted by launders into the respective extractors: No. 1 for strong solutions, No. 2 for weak solutions, and No. 3 for washings running to waste. The first liquor coming off from moist charges, or when sluicing is carried out, is run through No. 3, as it consists practically of the displaced water; but gradually the solution shows small amounts of cyanide, and becomes strong in gold. It is run first into No. 2 extractor, and then into No. 1. Generally, solutions above ·15 % would pass into No. 1; from ·15 to ·05 % into No. 2; below this, if necessary, into No. 3. But this last box is intended for occasional use only, and it is the practice to retain as much of the solution within the works as possible. This is accomplished, where the additional extractor is not provided, by limiting fresh water-washes to an amount equal to the moisture contained in the original charge—usually 15 to 20 %.

At the start of operations, the fresh zinc does not act as effectively as when in regular work, and a white hydrated precipitate of zinc cyanide is frequently formed when the solutions are very weak in cyanide. This precipitate disappears usually after a day or two, and may be prevented entirely by using stronger solutions at the commencement. There is not the same facility for regeneration of cyanide in the extractor at first; but as the solutions become more complex, matters speedily adjust themselves.

A large excess of lime used in neutralising acid ore has been found by Franklin White to cause a whitish precipitate on the zinc, which operates detrimentally in extraction.

In some American mills, the ordinary zinc box is replaced by light, rectangular sheet steel boxes, absolutely water-tight, and thoroughly coated with P. and B. paint; they are made by F. M. Davis, Denver, and closely resemble the charcoal box described further on. In service, these boxes are arranged in series, as shown in Fig. 170. Each box has 2 compartments, the solution passing down the narrow side and up the wide one (where the zinc shavings are packed), overflowing to the

next box. The shavings are confined between wire-screen plates, the top one being locked in place. Each box is provided with lugs for handle, and is easily manipulated ; capacity, 1 cub. ft. zinc shavings ; weight, 30 lb. ; price, 25*s.* The advantage of these separate cells is that at clean-up the desired number of cells can be taken out and replaced in a few minutes by other cells already packed, thus not delaying precipitation of gold solution ; and also that they can be cleaned quickly and thoroughly, and with less danger of loss in handling the gold mud.

When packing, the zinc shavings are placed in the com-

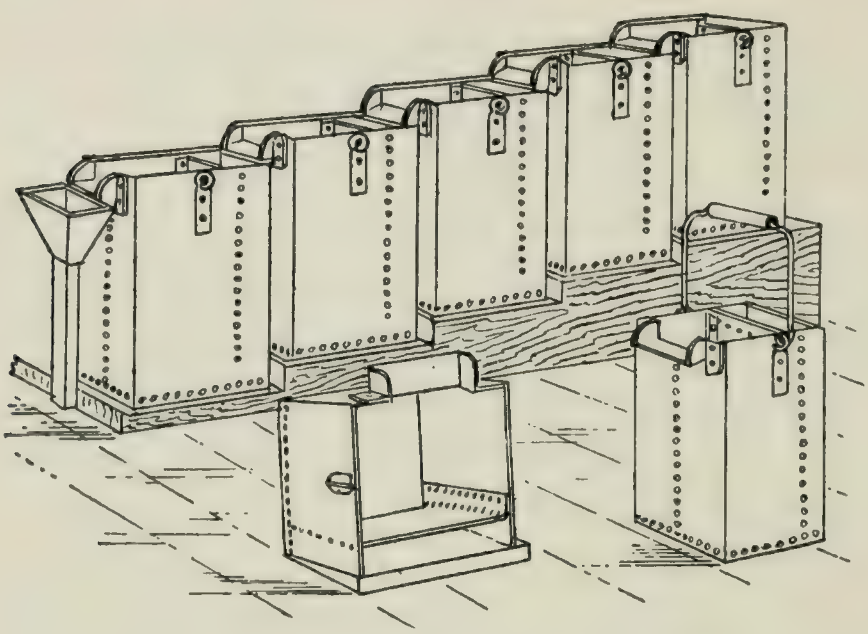


Fig. 170.—Precipitation Box.

partments or cells in layers, care being taken that each layer fits well into the corners and around the sides, as the solution is otherwise likely to form channels in these places ; the compartments are filled to within 2 in. of the overflow, and a small wooden strip is fastened across the top, so as to prevent the zinc from rising out of the solution if an appreciable amount of gas formed in the precipitation should accumulate.

Some 85 % of the values precipitated are deposited in the two first cells, or compartments of the box. In replenishing the zinc, that from the second compartment is put into the first, that from the third into the second, and so on, fresh zinc shavings being added in the last compartment. In this way, the values are practically all concentrated into the first two compartments.

Another recent innovation is a circular vat for containing the zinc shavings. The rectangular box has certain mechanical disadvantages, the most obvious being the tendency of the solution passing through it to travel up the sides, and particularly up the corners of the compartments, thereby forming channels. Other objections in practice are frequent liability, from faulty construction of boxes, of the solution to leak from one compartment to another without coming in proper contact with the zinc, and the inconveniences this leakage produces in cleaning-up.

It is urged by W. A. Caldecott * that a small vat obviates these drawbacks and at the same time provides accommo-

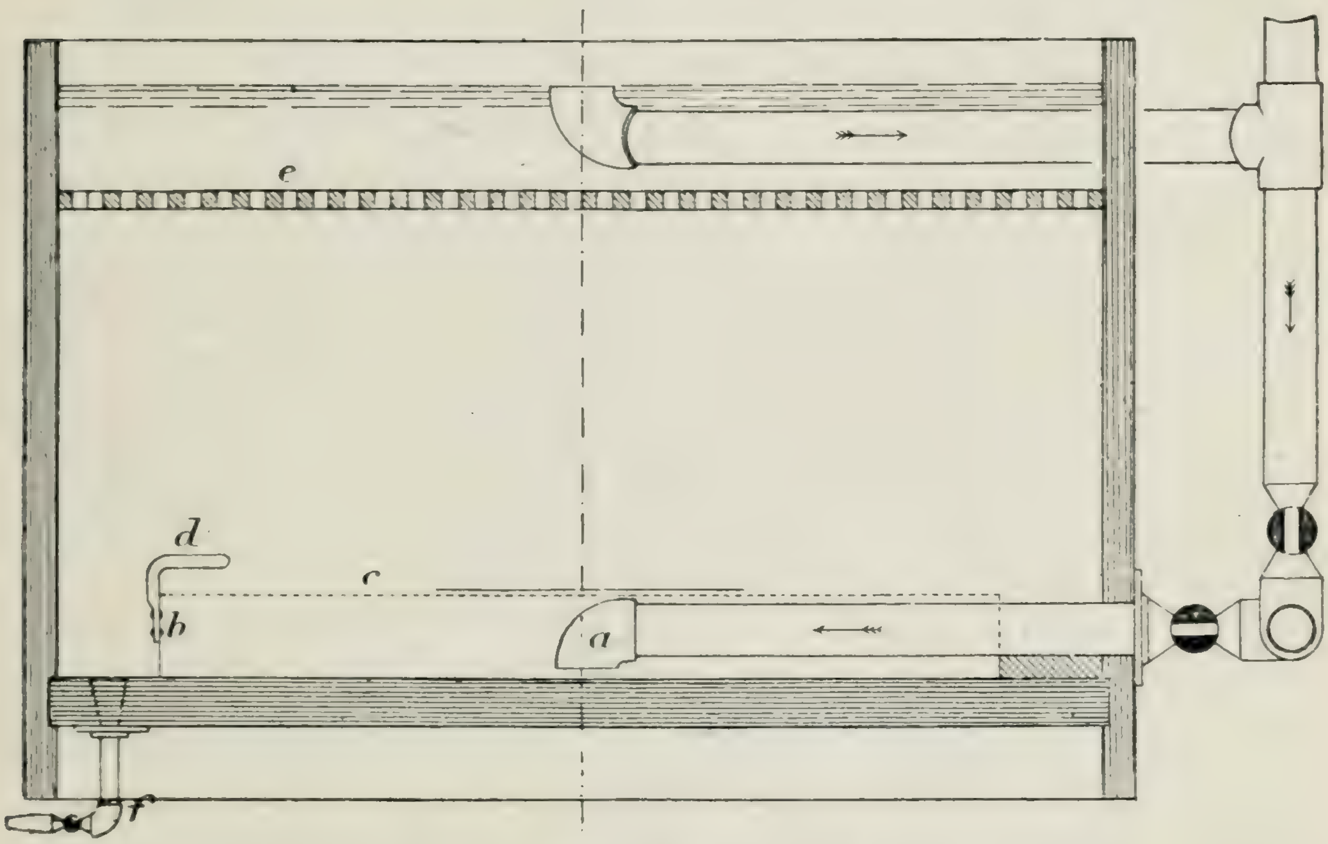



FIG. 171.—PRECIPITATION TANK.

dation for much more zinc per sq. ft. of floor space. Such a vat, Fig. 171, has been adopted by the Witwatersrand G.M. Co. with excellent results. It has an internal diameter of 5 ft. and depth of 3 ft., and may be arranged in a series of three; each vat contains about 40 cub. ft. of zinc. The even distribution of solution in the vat may be readily seen in practice from the uniform rising of the bubbles of gas to all

* Jl. Chem. & Met. Soc. S. Africa, Sept. 1899; En. & Min. Jl., Nov. 18, 1899.

parts of the surface. The method of introduction of solution at *a*, the packing of zinc between the steel ring *b* ($\frac{1}{16}$ in. × $4\frac{1}{2}$ in.) and the side of the vat, the large cross-sectional area of vat, and absence of corners, all tend to prevent formation of channels. For convenience in cleaning-up, the ring *b* to which is fastened sieving *c* ($\frac{1}{4}$-in. sheet iron with perforations) is made removable by handle *d*. The details of construction of these vats may be considerably modified, and a little consideration shows various ways of introducing solution at bottom and taking it off at top. For instance, the inflow pipe may be brought down the side of vat or introduced from the bottom, and in place of an elbow turned square down over the centre, may terminate in a ring of perforated pipe. Similarly, for convenience in cleaning-up, the bottom of the vat may be slightly conical or sloping to an outlet on one side, which delivers by a launder into a settling vat, or is connected with the pump of a filter-press. The arrangement of side pipe-line enables one vat to be cut out at a time without shutting off others, while cleaning up or dressing the zinc; and the open T-piece allows the vat to be readily emptied of solution, and likewise prevents accumulation of air or other gas at the angle. With a precipitation plant composed of small vats as units, the ease of re-arrangement or extension to any degree required is readily apparent. A wooden grating *e* serves as a cover for the zinc, and a $\frac{3}{4}$-in. drain is provided at *f*. Two sets of vats, of three in each series, have been employed in the precipitation of about 200 t. per day of weak (·04 %) cyanide solution, the rate of flow being 1 t. of solution per cub. ft. of zinc per $2\frac{1}{4}$ hours. The average value of solution entering was 3 dwt., and of solution leaving 2·5 gr., so that 96·5 % of the gold was precipitated.

It would seem that in operation some trouble arises from the leakage of valves, and the pipe arrangement shown has given place to a simple inlet and a decanting siphon.

But the latest form of precipitating box is an enamelled cast-iron deep pan, which is absolutely without action of any kind on the solutions, and can be kept scrupulously clean and easily emptied to the last trace. This is being introduced by

the Cyanide Plant Supply Co., and is likely to replace all other forms of zinc-shaving box in use.

Reactions.—It is found in practice that for every ounce of gold recovered, about ½ lb. zinc goes into solution. It has been generally accepted that this is due to the action of cyanide on zinc, and much has been written about the defects of zinc precipitation from the tremendous consumption of cyanide involved thereby. But Alfred James * made a number of tests, with results showing no appreciable difference between the strength of solutions in active cyanide when entering and leaving the extractors. He suggests that whilst there is probably a certain amount of reaction between the auro-cyanide and zinc, the greater reaction is between the zinc and the caustic potash produced, with liberation of hydrogen ; and that the nascent hydrogen, reacting on the auro-cyanide, regenerates the cyanide and precipitates the gold. Moreover, if any gold be present as auri-cyanic acid, it is precipitated by the same means, so that precipitation is actually more perfect than would be afforded by zinc alone, "which will not effectually precipitate gold from a solution of auri-potassic cyanide."

That the reaction is really one between caustic soda and zinc rather than between cyanide and zinc is corroborated in a very striking manner by certain published accounts of the behaviour of copper when present in ordinary dilute works solution. It is found that the copper is precipitated at such a rate as to coat the zinc, and to hinder precipitation thereon of the gold ; but that an addition of cyanide to the solution, so as to strengthen it before it enters the extractors, quite overcomes this difficulty. This is explained by the assertion that the copper remains in solution in consequence of addition of cyanide, gold only being precipitated. But James is of opinion that the action with the zinc becomes more vigorous, the copper being precipitated in flakes instead of as a varnish, and therefore not interfering with the action of the zinc on the caustic soda or gold.

Or the conditions may be accounted for by the electromotive questions involved. The difference in E.M.F. of a

* Trans. Inst. M. & M., iii. 400 (1895).

zinc-copper couple in solutions of KHO and KCy is very marked ; in KHO, copper and gold are very negative to zinc and to almost the same extent, and therefore the two metals would be precipitated together from solution roughly in proportion to the comparative amounts present, and copper-coated zinc would have practically no action on any gold in solution. Whereas, in cyanide solutions, copper is only slightly negative to zinc, and exceedingly positive to gold, which would therefore be precipitated practically copper-free, unless the latter were present in considerable excess ; and even copper-coated zinc would still be effective.

James explains the reactions necessary for the whole process thus :—

$$2Au + 4KCy + H_2O + O = 2KAuCy_2 + 2KHO$$

and

$$2KAuCy_2 + 2KHO + Zn = Au_2 + 2KCy + 2HCy + ZnK_2O_2$$

but does not deduce from this equation that 6 oz. gold ought to be extracted at a cost of 1 oz. zinc, as $ZnK_2O_2$ combines to a certain extent with HCy and KCy to form zinco-potassic cyanide—

$$ZnK_2O_2 + 2KCy + 2HCy = ZnK_2Cy_4 + 2KHO,$$

so that an ultimate product of the re-action is zinco-potassic cyanide and caustic potash. This zinco-potassic cyanide has itself a solvent action on gold, and extractions have been made from ores by employing it as the dissolving agent. It has also been pointed out by Feldtmann that it may be converted into cyanide of potassium and sulphide of zinc, by re-action with potassium bi-sulphide produced during leaching of the ore—

$$K_2ZnCy_4 + K_2S = ZnS + 4KCy,$$

or by iron sulphide—

$$K_2ZnCy_4 + FeS + 2KCy = ZnS + K_4FeCy_6\,;$$

any zincate of potash would also be similarly converted into sulphide of zinc and potassium hydroxide—

$$K_2ZnO_2 + K_2S + 2H_2O = ZnS + 4KHO.$$

The zinc is thus prevented from accumulating in solution.

Lead-Zinc.—The question of purity of zinc for precipitation purposes has been very much argued among S. African operators, many of them claiming that zinc containing 1 to 2 % lead is more effective than the pure article. On the other hand, it is contended that lead is not beneficial or necessary in hastening precipitation, and that it introduces a great drawback, viz. that the bullion is thereby fouled, and that the refining needed to produce high standard bullion is made much more difficult and costly. Opinions remain divided on the point.

While in some quarters condemnation is given to the use of leady zinc, where the combination is in the nature of an alloy, at the same time there is approval of zinc shavings on which a superficial coating of lead has been deposited, and it is asserted that the two forms of leaded zinc are not identical in action. It has become the practice at several Transvaal mills to produce a zinc-lead couple, by immersing the zinc shavings in a 10 % solution of lead acetate, thoroughly washing and stirring them until they attain a dark colour. It is found advisable that such prepared shavings should be placed in the zinc box and covered with gold solution as quickly as possible, since exposure to the air is detrimental to their efficiency.

At the Crown Deep slimes plant, the zinc box used is 40 ft. long, 6 ft. wide, and 5 ft. deep, divided into 9 compartments. The auro-cyanide solution flows through it at the rate of 15 t. per hour. An essential part of the method there introduced by T. L. Carter and W. K. Betty,* besides the lead-coated zinc as already described, is the addition, at the head of the box, when commencing precipitation, of 2·5 % solution of fresh cyanide, which is allowed to run freely into the gold liquors for about 4 hours, bringing their cyanide strength from ·007 up to ·025 %. When 20 lb. of KCy has thus been introduced, a further 10 lb. is admitted during the next 6 hours, increasing the strength yet again by ·007 %. After 12 to 14 hours, only a slow drip of strong solution into the gold liquor is maintained, so as to raise the contents of

* Jl. Chem. & Met. Soc. S. Africa, Oct. and Dec. 1898.

KCy in the liquor undergoing precipitation by about ·007 to ·008 %, and this is continued to the end of the run of that box. During the first 4 or 5 days, precipitation is very active; but gradually the zinc in the upper compartments loses its power, and, after 10 or 11 days, becomes "dead" and rises to the surface, though meantime there is no diminution of energy in succeeding compartments. When the zinc is thus allowed to become "dead," its dissolution causes much trouble during the clean-up. The evil has been remedied by running the zinc only for 2 or 3 days in the slimes-plant precipitators, and then transferring it to the sands plant, where it remains serviceable under the influence of the stronger KCy solutions therein employed. Fresh zinc is placed in the slimes-plant extractors to restore the balance. The cost of precipitating the gold liquors from 9295 t. of slimes in this way, in October and November 1898, is given as follows:—

| | *d.* |
|---|---|
| Cyanide | 4·05 |
| Zinc shavings, including cutting | ·66 |
| Sulphuric acid | ·61 |
| Power | ·31 |
| Lead acetate | ·22 |
| Total | 5·85*d.* per ton. |

The effluent liquor from the boxes carries about 2 gr. gold per ton.

After mature consideration, this process was adopted at the Robinson mill in 1898, and the improvement in extraction from 72·1 % to 77·2 % is largely attributed, in the official report, to this circumstance.

At the Ulundi mill, Transvaal, A. H. Hartley* had most satisfactory results with ore containing "a considerable percentage of copper." On 5 to 8-dwt. tailings giving gold liquors of 10 to 39 dwt. per ton, in ½ to ¾ hour's contact (1 to 1½ t. per cub. ft. of zinc per 24 hours), there was "almost complete precipitation of the gold whether strengthened or not before entering." Liquors at 20 dwt. and only ·009 % KCN were freed from gold, and the cyanide consumption was

* Jl. Chem. & Met. Soc. S. Africa, May 1899.

reduced to ·5 lb. per ton. But even with ordinary zinc the precipitation was practically as perfect, and this is attributed by Hartley to the presence of copper. (See p. 611.)

Zinc fume.—Notwithstanding admitted drawbacks in the use of zinc shavings, the principal one being that precipitation can never be made complete by percolation, because of the difficulty of ensuring intimate contact between the zinc and the gold carried in solution, yet zinc, being the cheapest of the metals which are electro-positive to gold in cyanide solutions, remains in favour. In another form, however, namely as zinc fume or dust, absolute and intimate contact can be rapidly and completely secured by agitation. Moreover, zinc fume is a commercial by-product made in great quantity during the preparation of the metal from its ores by the usual distillation process, being the condensed metallic vapour in a state of extreme division ; 95 %, or even more, will freely pass a mesh of 200 holes to the linear inch, so that is in fact an impalpable metallic flour. Under the microscope, it is seen to consist of spheres more or less coated with oxide. This coating renders the action of the untreated product somewhat slow as a gold precipitant ; but on shaking with a dilute solution of ammonia, ammonium chloride, or carbonate, or with any other suitable solvent of zinc oxide, and stirring the deposited fume with water, a metallic emulsion is produced, which exhibits remarkable gold-precipitating power. A small portion agitated with very dilute solution of auro-potassic cyanide (with or without the presence of free potassic cyanide) permeates the whole volume of liquor, rendering it slightly and uniformly cloudy. The minute suspended metallic particles pervade the solution, partly dissolve in it by replacing the gold in the auro-potassic-cyanide, partly act as nuclei, and almost immediately determine precipitation of the gold. In a minute or two, gold films grow around and upon these minute nuclei, and a precipitate begins to agglomerate and fall out in increasing masses, so that within 5 minutes the whole of the precious metal is collected at the bottom of the vessel as a more or less black powder (according to amount of fume used) of very granular nature. Owing to

the extreme fineness of its particles, the zinc can be rapidly brought into almost molecular contact with the dissolved gold ; by regulating the quantity of precipitant to the work to be done, it need never be in large excess at any moment, and considerable economy is thus effected. Moreover, precipitation is practically absolute, and the gold is thrown down in quite a different condition from the mud produced in zinc-shaving boxes—it is a heavy blackish granular agglomerate ; and though shaken up with liquor repeatedly, it falls again completely in a few seconds. The precipitate contains much less free zinc, and no large fragments of that metal.

Its use has been strongly advocated by Sulman,* who found no difficulty in getting from strong gold solutions (as obtained from tailings by the use of ·3 % cyanide) and the employment of only 3 times the weight of zinc fume compared with the quantity of gold shown by assay, gold precipitates of over 600 fine (the balance of the zinc being dissolved during precipitation), whilst by washing the granular powder with a little dilute sulphuric acid, a fineness of 900 can be reached. It is capable of automatic application to the continuous flow of liquors derived from the leaching tanks ; while on the score of cost it is much superior to shavings, being only about half the price, and often less, and the consumption per oz. of gold precipitated being greatly diminished.

It is difficult to understand why zinc fume has not come into more extended use. In the author's experience with it in trial runs on gold liquors from 10-dwt. slimes, using weak KCy solutions, and without fortifying the liquors, the results were eminently satisfactory. The liquors were drawn by a steam ejector though a sand filter to clarify them (p. 601) into an agitator. This was a cylindrical steam-jacketed vessel with rotating beaters, that had been previously used for washing kerosene shale oil. The weight of zinc fume estimated to be necessary for the gold contents as shown by assay was admitted through a trap in the lid, and thorough agitation was given for 20 to 30 min. The whole volume of liquor and gold-zinc mud was then discharged into a small centrifugal

* Trans. Inst. M. & M., iii. 222 (1895).

lined with stout burlap, the precaution being taken to run the effluent liquors through charcoal filters to the sump, in case of any possible mishap or leakage. No trace of gold could be found in the effluent. The steam jacket was adopted as a safeguard against very low air temperatures, which were liable to be encountered at short notice ; but it was not used. No supply of zinc fume being available in Australia, extended runs could not be made with it ; but so far as trials went, it was by a long way the most satisfactory and economical of all precipitants.

So far as the author knows, the only two large works using zinc fume are at Mercur (Utah) and Deloro (Canada). In De la Mar's Mercur mine, according to H. L. J. Warren,* precipitation by zinc dust, combined with stirring and agitation by compressed air, is a "patent process controlled by Captain De la Mar," and gives excellent results. Perhaps the limited use of the article is in a measure due to the disinclination to pay royalty to a patentee for the right to employ means which would seem to be free to all the world. One of the curses of the cyanide process has been the feverish rush to patent every possible detail of mechanical plant and chemical reagent, notwithstanding that in almost every case they are direct copies of or adaptations from methods long known and applied in other branches of chemical engineering.

The exact mode of procedure at Mercur is as follows. The auro-cyanide liquor is pumped into 3 precipitation tanks, each 14 ft. diam. and 8 ft. deep, and is kept constantly agitated by air under 20 lb. pressure forced in through a ½-in. iron pipe. When the tank is first filled, 30 lb. zinc dust are put in ; on the second charge, 20 lb. ; on the third and successive charges till the end of the monthly run, 10 lb. After the first charge, no zinc is added till the tank is about half full, when ¼ of the total charge for the tank is put in, and the remainder as the tank fills. Each full charge of liquor is 30 t. The solution is allowed to settle for an hour after precipitation is reckoned to be complete, and the supernatant liquid is drawn off through filter-presses to arrest any suspended particles. The

* En. & Min. Jl., Dec. 30, 1899.

exhausted liquor, which is not lost, but goes to a sump 24 × 8 ft., is said to rarely exceed 20 *c.* per ton, which is equivalent to 5 gr. fine gold in each ton of spent liquor, the

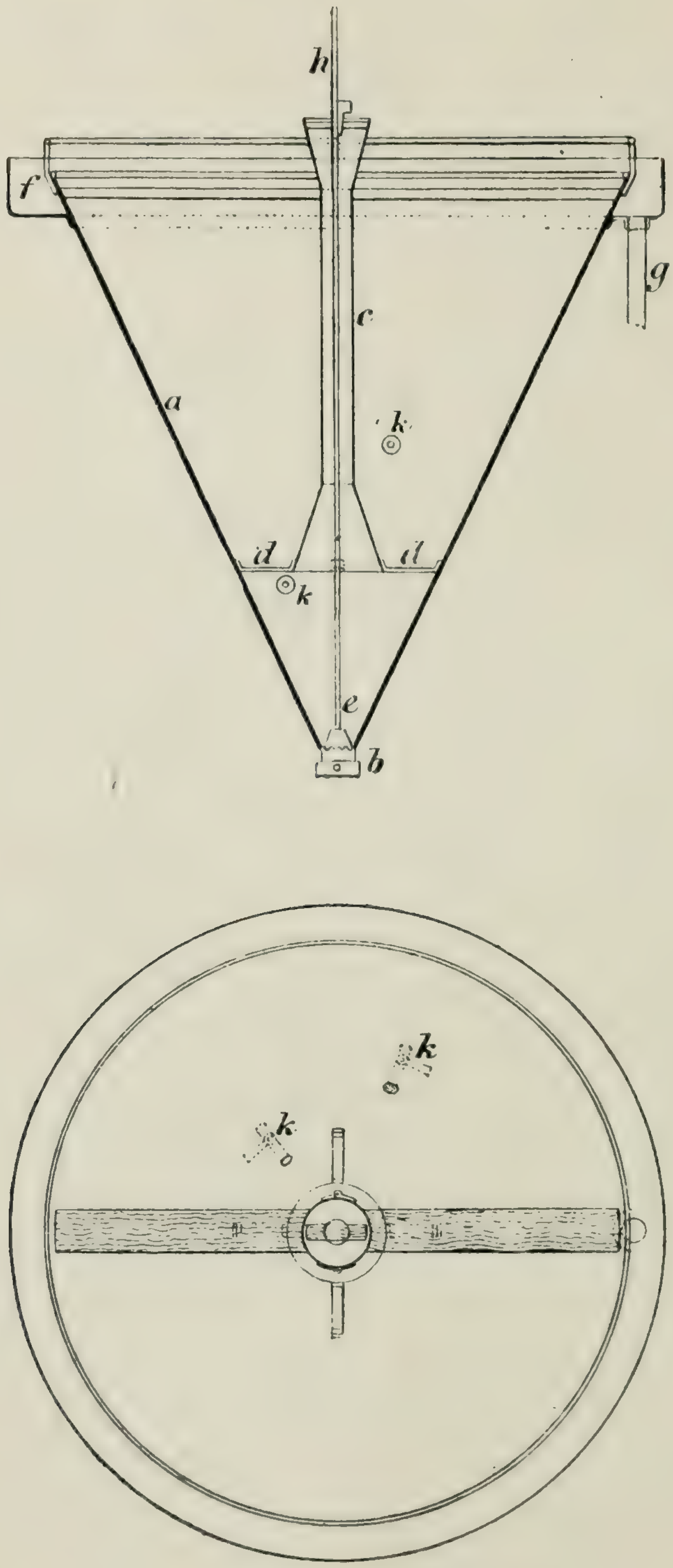



FIG. 172.—PRECIPITATION CONES.

original auro-cyanide solution having contained 3 dwt. This means rather more than 93 % precipitation, and can only be

considered moderately good work. The amount of zinc consumed per oz. of gold precipitated is $1\frac{1}{3}$ lb., which again does not compare well with S. African practice and zinc shavings. There are probably two reasons why better results are not attained, the first being that no pains are taken to eliminate zinc oxide from the zinc fume, so that precipitation is rendered much slower than it should be ; and the second, that pneumatic agitation is not thorough, and cannot be counted on to ensure complete contact between the solid precipitant and the gold in solution. Apparently acidity is not to blame, for it is the custom at these works to add 100 lb. caustic soda to each 100 t. of solution before standardising with KCy ; and it is reported that this aids precipitation by causing a more vigorous action of the solution on the zinc dust, increasing precipitation and decreasing consumption of precipitant.

At Deloro, the zinc fume precipitation plant embraces the arrangement designed by H. L. Sulman in connection with his bromo-cyanide process. It consists of a sheet-iron funnel vat *a* (Fig. 172), about 5 ft. diam. and 5 ft. deep, having at the apex a 3-way valve *b*, which serves to admit liquor or to discharge precipitate. Zinc fume is introduced to the liquor by way of the central pipe *c*, which is expanded at the lower end, and has riveted to it carriers *d* which are free of the sides of the vat *a*. Immediately above the inlet *b* is a rose *e* (Fig. 173) for distributing the stream of liquor as it enters, so as to facilitate commingling of liquor and fume. Around the vat top is a launder *f* and discharge pipe *g* for conveying away liquor as it overflows, the circulation being continuous. The apex inlet is closed by a wooden rod *h* carrying a rubber-faced plug *i*. Pettcocks are provided at *k*, for drawing off final liquor without disturbing the precipitate.

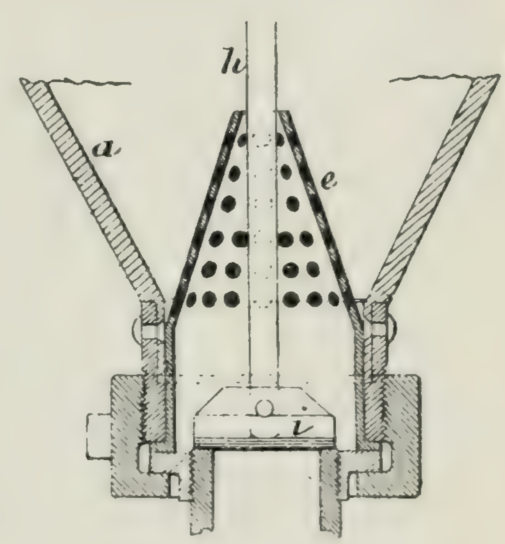



FIG. 173.—PLUG.

The vats are made of galvanised iron, or if of plain iron

are covered with a coating of zinc-fume paint in boiled linseed oil.

The liquors are pumped up from the sump tanks into an overhead reservoir which delivers a stream at constant pressure to the precipitators. The conical shape of the latter reduces the velocity of the rising stream, so that the undecomposed zinc and precipitated gold almost completely sink to the bottom of the vat. To arrest any escaping particles, the stream is passed into the bottom of a settling box $8 \times 1\frac{1}{2} \times 2$ ft. deep, carrying a number of transverse slabs (of smooth wood, slate, or glass), inclined at $45^{\circ}$ and placed about $\frac{1}{2}$ in. apart. But the amount so collected is very small.

Zinc fume is used in the proportion of 19 lb. for 15 t. of strong liquor containing 90 oz. gold, and is admitted as an emulsion. About 12 ib. are introduced as the liquor commences to enter, $1\frac{1}{2}$ lb. when 3 t. have passed in, $1\frac{1}{2}$ lb. after 6 t., 2 lb. after 9 t., and 2 lb. after 12 t.

As soon as all the liquor has been admitted, the bulk of the precipitate or gold-zinc mud may be drawn off. The rest is left to settle for 2 hours or so, and the clear liquor is then run out and pumped into the first portion. Any remaining precipitate is carefully rinsed out and added to the previous collection.

Weak liquors are precipitated in the same way, but the zinc fume is introduced in smaller quantities at shorter intervals. For 14 t. weak liquor, 17 lb. zinc fume are used.

The total consumption of zinc fume is about ·2 lb. per ton of ore assaying about 15 dwt. Precipitation is not carried to an extreme point, as the liquors are used over and over again; below 2 dwt., the consumption of zinc is disproportionate to the results attained.

It is only fair to say that this method is entirely free from patent restrictions.

For estimating metallic zinc in zinc fume, the course proposed by Horsin Deon is to use a Fehling solution prepared according to Soxhlet's method, in which the two liquors are kept separate; 34·64 grm. copper sulphate, containing 8·785 grm. copper, will be reduced by 9·178 grm.

zinc. The copper not precipitated is determined by solution of glucose.

*Lead.*—Electrolytic precipitation on lead cathodes, known as the Siemens-Halske process, has been on trial for about 6 years in the Transvaal. It is very fully described by M. Eissler,* who prefaces his description with some general remarks on electrolytic action. Referring to the decomposition of a solution of a metallic salt by the electric current, with consequent deposition of the metal on the negative pole and liberation of the metalloid at the positive pole of the cell, he remarks that the law holds good only for solutions strong in metal; with very dilute solutions, as used in the cyanide process, the current does not find sufficient of the metallic compound present at the electrodes, wherefore decomposition of water also takes place. For this reason, to get efficient precipitation, constant diffusion of the solution is requisite, "and the most economical and convenient way of effecting this is to allow a slow but steady flow through the precipitation boxes. But it is still more important to give a very large surface to the electrodes. In fact, a better effect is obtained by doubling the number of plates than by increasing the current tenfold."

It follows that desirable conditions in the cathode are:—

(*a*) Capability of being presented in very thin sheets, to afford maximum area at minimum weight.

(*b*) Power to retain the precipitated gold by adherence.

(*c*) Ready release of the gold.

(*d*) Inferiority to the anode in electro-positiveness, so that no return current may be generated.

These conditions are fulfilled by lead, which is rolled into very thin sheets, fastened (3 in each) in light wooden frames 2 × 3 ft., giving in each frame a total area of 36 ft.; thus 87 frames in a "box" afford 3132 sq. ft., and each sheet weighing 1 lb., the box carries 261 lb. of lead.

The anode used is iron, which causes formation of Prussian blue by reaction between the oxide of iron and ferrocyanide; from this, the cyanide is recoverable by dissolving in caustic

* Trans. Inst. M. & M., iii. 77 (1895).

soda, evaporating, and smelting residue with potassium carbonate. But a considerable amount of gold accompanies this Prussian blue, and its recovery is in the nature of a by-product, which is not satisfactory.

The current necessary is estimated at about ·06 ampere per sq. ft. ; and with cathodes 1½ in. apart, 7 volts suffice. A weak current gives hard gold deposits and minimises waste of anodes. Any accident to the electric connections—a constant source of worry in all electrolytic methods—means an immediate reversal of current, with re-dissolution of the precipitated gold.

Power is computed at about 2 watts per ton, or (at 746 watts = 1 h.p.) approximately 1 h.p. per 500-t. plant.

The precipitation plant on this system erected at the Worcester mill, Transvaal, consists of 4 boxes 20 × 8 × ·4 ft. deep. Copper wires are fixed along the top of the sides of the boxes, and convey the current from the dynamo to the electrodes. The anodes are iron plates 7 × 3 ft. × ⅛ in. thick. They stand on wooden strips in the bottom of the box, and are kept vertical by wooden strips fixed to its sides ; some rest on the bottom, while others are raised about 1 in. above the level of the solution, thus forming a series of compartments similar to those of a zinc precipitation box, the difference being that the solution passes alternately up and down through successive compartments. The plates are covered with canvas, to prevent short circuits. The lead sheets are stretched between two iron wires, fixed in a light wooden frame, which is suspended between the iron plates. The boxes are kept locked, being opened once a month for the clean-up.

Sometimes lead shavings are packed between the iron plates, in place of lead sheets, securing an economy of metal and an increase of area exposed.

The bullion is collected in the lead, which contains 2 to 12 %, and is recovered by cupelling, the resulting litharge being available for reduction to metal and re-use.

Published results of the use of the Siemens-Halske process exhibit wide differences, but in no case do they indicate that it possesses remarkable advantages over either zinc or charcoal.

At the Crown Deep, according to T. L. Carter,* on 10,317 t. of slimes in April–June 1898, the cost was :—Cyanide, 6·96*d.* per ton ; lead-foil (less litharge recovered), 3·84*d.* ; power, ·95*d.* ; royalty, 1·18*d.* ; total, 12·95*d.* per ton, or double the cost of treatment by leady zinc.

The Crown Reef slimes plant,† from July 1896 to October 1898, treated 82,542 t. for a theoretical extraction of 74·6 %, while the fine gold actually recovered from lead bullion and by-products only reached 54·4 %. In order to explain the discrepancy, the furnace in which the lead was melted was torn down and ground up in a ball mill ; the sides of the precipitation boxes were planed and the shavings were burned ; the anode sacking was removed and burned ; but still the difference of 20 % could not be explained. Every precaution, supported by ample means, was taken to check the tonnage, which is usually one of the main sources of error. The conclusion is arrived at that there are too many ways in which gold can escape by this process to make it permanently satisfactory. If the bullion won had been recovered in the form of zinc bullion, the company would have received 2644*l.* more for it ; or 932 oz. less gold might have been recovered, and the company would still have earned the same amount of money. The gold derived from by-products was 25 % of the total recovery.

At the Ferreira, the theoretical extraction is 80·18 % and the actual is 76·09 %. The consumption of lead is 1 lb. per ton of slime. The flow through the boxes is 15 to 20 t. an hour, the solution entering at 1 dwt. 5 gr. and leaving at 4 gr. per ton. By-products are treated in a pan furnace and cupellation furnace at a cost of about 2·5*d.* per oz. of gold.

The Bonanza mill gets 81·15% actual on 82·72% theoretical extraction.

A trial run at the Nourse Deep in 1899 gave a return which contrasted very unfavourably with those from the zinc-lead couple.

The official report of Glynn's Lydenburg for 1899 says,

* Jl. Chem. & Met. Soc. S. Africa, Oct. & Dec. 1898.

† S. J. Jennings, Mineral Industry for 1898, p. 333.

" During this, as in the previous year, continued trouble has been experienced with the electric precipitation process," and instead the ordinary zinc process had been installed and was working most satisfactorily.

*Iron.*—The idea of using an iron cathode was suggested (and patented) by the well-known electro-metallurgist E. M. H. Andreoli. The anodes employed are plates of peroxidised lead. Removal of gold deposited on the iron cathodes is effected by immersion in molten lead. While there is perhaps less contamination of solutions in this method than in zinc or lead precipitation, it is somewhat roundabout and inconvenient; and though the patent rights were purchased by the Rand Central Ore Reduction Co., nothing has appeared in print to show that the process has ever been installed on a working scale.

*Charcoal.*—The use of wood charcoal as a precipitant from cyanide gold liquors as well as from chlorine gold liquors (p. 456) is very general in Australia, notably in the colony of Victoria, where plants for dealing with tailings are quite numerous. One of the most advanced, perhaps, is that at the South German mill, Maldon, which has been several times described.*

The extractor house of a plant dealing with 2400 t. a month contains 198 wooden tubs, each 2 ft. 4 in. high by 2 ft. 1 in. diam. at top and 1 ft. 9 in. at bottom. In the centre of the bottom of each tub are two small wooden cleats, 4 in. apart, on which rests a glazed drain pipe, 4 in. diam., flanged end downwards. The centre pipes and the tubs are filled with charcoal, coarse for 6 in. from the bottom, and medium size to within 6 in. of the top of the centre pipe, with about 3 in. outside and 5 in. inside the pipe of coarse charcoal again. On the top of the charcoal, all round the pipe, is a wooden lid weighted with flints, to keep the charcoal from floating and blocking up the outlet pipe.

Another form of filter tub much used has no centre pipe, but there is a false bottom covered with a filter cloth, above

* J. Mactear, Trans. Inst. M. & M., vi. 43 (1897); J. I. Lowles, *ibid.*, vii. 190 (1898); W. B. Gray, Trans. Aust. Inst. Min. Engs., v. 138 (1898).

which is charcoal, and on top a hinged lid, kept locked, with the feed pipe let into it. The solution leaves the tub under the false bottom by means of a siphon pipe to the next tub.

The preparation of the charcoal is important. Fine stuff is objectionable, as it will pass through into the sumps with the solutions, besides making a bad filtering medium when closely packed. Hard compact charcoal, again, is not nearly so efficient as that which is soft and porous. Gold is deposited throughout the whole of light charcoal, so that compact material offers far less surface. Moreover, it would take longer to incinerate. The charcoal is crushed in a Dunn mill and sieved to three sizes, coarse (about $\frac{1}{2}$ in.), medium, and fine, the last being rejected. The other sizes are thrown into separate tanks of water, and washed ; the compact coal sinks, the light, porous stuff of both sizes is skimmed off, and allowed to dry before being used. About 30 % of the original quantity of charcoal is lost by this treatment.

The tubs are arranged in two groups, the first of 144 filters, in 24 sets of 6 each, for strong solution ; and the second of 54 filters, in 18 sets of 3 each, for alkaline wash (containing ·04 % KCy) and weak solution. The solution, passing through a 1-in. branch from a 3-in. main, travels down the centre pipe of the top filter in each set, coming up on the outside, over the lid, and through another 1-in. pipe to the second filter, and so on, finally passing away by a launder to its own particular sump. The top of each of the last 5 filters of a set is placed 3 in. lower than the top of the preceding one. The top filter naturally catches most gold, and when fairly charged, is removed, and each of the remaining 5 tubs of the set is moved up one step, bringing second filter into first place, a tub with fresh charcoal being put into sixth position. The filter removed is relieved of some of its solution (to save loss during moving), and emptied into a large box with a filter bottom, when what solution there is in the tub drains away to the strong-solution sump, leaving the auriferous charcoal ready to be shovelled out and taken to the furnace room. Every day 8 top strong solution filters are emptied in this way, so that a tub occupies first position during 3 days. The flow of solution

from the intermediate tanks to the filters is regulated by valves in the main, and the following ingenious arrangement prevents that flow being exceeded :—At the junction of the main with the distributing pipe, a small open glass tube is inserted, in which the solution under the right pressure rises to a certain height ; very slight excess over the proper pressure will cause this tube to overflow, obliging the attendant to pay strict attention to the amount of solution he allows to pass through. The launders from the filters are connected with all the sumps, so that, if necessary, any one solution can be passed over the whole of the filters ; but, as a rule, the filters are worked in two groups, and then about 400 gal. of strong solution per hour pass through 144 tubs, and 300 gal. of weak or alkaline through 54 tubs.

The first filter in a set catches 45 %, the second 25 %, the third 15 %, the fourth 9 %, and the fifth 5 %. Strength of solution does not seem to affect percentage of precipitation at all ; effluent liquors going out into the launders from the tubs rarely show more than 2 to $3\frac{1}{2}$ gr. of gold per ton, and solutions down to ·04 % KCy are successfully precipitated from.

The gold caught by the top filter of a set is considerably purer than that caught in the lower ones, steady diminution in percentage of gold taking place downwards through a set, till in the bottom tub the bullion is nearly pure silver. However, as the filters are shifted from the bottom upwards in rotation, each catches about the same ratio of gold and silver in the end. This might be utilised, however, to produce higher standard bullion in the first row or two.

The main drawback to the process is the enormous bulk of precipitate and precipitant to be dealt with at clean-up, and the risks of mechanical loss incidental thereto. Labour required in preparing charcoal and manipulating tubs and pipes is quite a considerable item, but probably does not exceed that involved in making zinc shavings and manipulating the boxes. Outlay for plant is higher, each tub costing about 25*s.*, so that an installation of 198 will amount to nearly 250*l.* as against 60*l.* for zinc boxes of equal working

capacity. Floor space also is much greater. James computes that 45 cub. ft. zinc shavings will do the same work as the 144 tubs (720 cub. ft.) of charcoal on strong liquors, and 34 cub. ft. as the 54 tubs (270 cub. ft.) on weak liquors.

The tubs and pipes may be replaced by thin wrought-iron boxes made as shown in Fig. 174.

The partition *a*, riveted to two opposite sides of the box, is carried down to within $1\frac{1}{2}$ in. of the bottom, dividing the

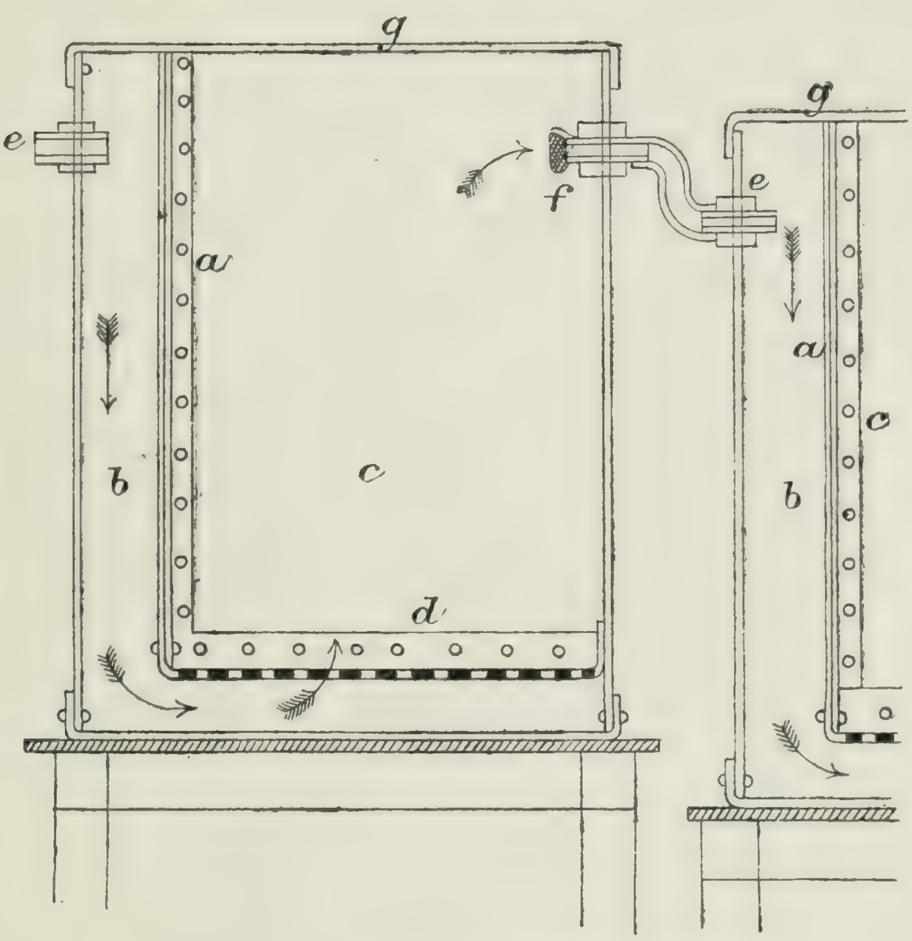



FIG. 174.—PRECIPITATION BOX.

box into a narrow outer compartment *b* and a wide inner compartment *c*; the former is simply a conduit for downward flow of solutions, while the latter is filled with charcoal through which the solutions must rise. The partition helps to support a false bottom *d* of perforated iron or stout wire mesh, about $1\frac{1}{2}$ in. above the true bottom, for carrying the charcoal. The influx of solution is at *e* and the efflux at *f*. The pipes and rubber connection tubes, as shown, are somewhat troublesome, and may be replaced by lips, a small orifice being cut in the sheet-iron lid *g* for the intake. In this way, much less fall may be provided, with advantage in rendering the flow very gradual.

Precipitation with this system at Lucknow was perfect, effluent liquors giving no reaction for gold. The author's intention was to use charcoal only for the weak liquors and as a safeguard for final effluents, thus very much reducing labour for renewal of charges, etc. Zinc dust and rapid recovery by agitation and centrifugal filtration were designed for the strong solutions.

According to Dr. Gaze,* the Australian hardwoods do not make at all suitable charcoal. At Lucknow, the charcoal was precisely the same as that employed in the smithies, being burned from box, stringy bark, red gum, and other eucalypts, and no difficulty was experienced. Similarly, the author was assured at Maldon, Walhalla, and other Victorian centres using charcoal, that no pains were needed to select certain woods, as those ordinarily chosen by the charcoal burners are quite satisfactory. Soft wood charcoal, in fact, possesses a drawback in that it readily crushes to dust, which is a serious matter in several ways.

It is also asserted by Dr. Gaze (p. 127) that charcoal becomes "sickened," so that "when 1 cwt. contains 10 oz. gold it ceases to act efficiently," and that it can be revivified and made to absorb 30 to 40 oz. by reburning in a closed vessel without access of air.

As to cost, Lowles places it at 4·29*d.* per oz. of gold, or ·545*d.* per ton of ore treated, with charcoal at 9*d.* per 40 lb.; as against zinc, 2·25*d.* per oz. of gold or ·275*d.* per ton treated, with zinc at 4½*d.* per lb. Labour is also in excess. Gray puts the cost of charcoal at 2*s.* per filter, or 2·75*d.* per oz. of gold saved, as against zinc at least 1 lb. at 6*d.* per lb. So far as Australian figures go, Gray is distinctly more correct than Lowles. At Lucknow, New South Wales, zinc shavings in 1899 cost the author 6·85*d.* per lb. on the mine, while charcoal of excellent quality was delivered at 1*s.* per bag of 60 lb.

While its bulk and unhandiness in manipulation are certainly drawbacks, it would seem that charcoal may be used with considerable advantage in timbered countries, and especially for weak solutions and effluent liquors, being

* 'Practical Cyanide Operations,' p. 124 (1898).

effective on all strengths, and regardless of KCy, alkalinity, or acidity in solutions.

*Aluminium.*—This metal has attracted much attention from experimenters on electrolytic methods of precipitation, because it is so electro-positive to gold in cyanide solutions that it will precipitate the precious metal whether any external galvanic current be maintained or not, whereas in other electro-deposition processes any interference with the electric connections leads to a reversal of current and re-solution of the precipitated gold. Moldenhauer in 1893 proposed to replace zinc shavings by aluminium sponge or shavings in the ordinary box, and some experiments on those lines were made in the Transvaal, but never came to anything. More recently, S. Cowper-Coles * has advocated the substitution of aluminium cathodes for the lead as used in the Siemens-Halske process, taking advantage of the readiness with which the precipitated gold can be stripped clean from the plates. He finds that it also has the advantage of reducing the cost of labour and economising the amount of KCy used, as the solution is not contaminated by any base metal, such as zinc. The deposition of gold from a cyanide bath on to an aluminium plate proceeds in a uniform manner, but in such a way that the gold is deposited as a metallic sheet, which is easily detachable from the cathode by stripping or rubbing, almost as soon as it is formed, and permitting direct and continuous recovery of the gold. Experimentally, it has been worked successfully on cyanide solutions containing only ·01 % KCy and $2\frac{1}{2}$ dwt. of gold to the ton of solution. The best results are obtained when the solution is raised to a temperature of about 100° F. It is also found advantageous to use a greater current density, with the necessary increased voltage, when the aluminium plates have been freshly placed in the solution, so as to ensure their being covered with a film of gold as quickly as possible, otherwise there is a tendency for aluminium hydrate to be formed. In about 10 hours, an extraction of 95 % was obtained by Cowper-Coles, on aluminium cathodes, the amount of gold in solution before electrolysis being $2\frac{1}{2}$ dwt. to the ton of solution, the

* Trans. Inst. M. & M., vi. 219 (1898).

strength of cyanide ·01 %, and the rate of flow about 15 gal. per 100 hours for every cubic foot of electrolysing cell, or 3 sq. ft. of cathode service. Gold has been deposited on similar cathodes from solutions containing only ·0075 % KCy, the current density being ·03 ampere per sq. ft., with 6 volts E.M.F at the poles of the generator.

Nothwithstanding the facts that aluminium does not vitiate the bullion produced, that it forms no cyanide combination, and that, theoretically at any rate, as soon as the gold is set free, a regeneration of potassium and sodium cyanides takes place in the liquor, so that the liquor leaving the extractor is more potent as a gold solvent than when it entered—all being considerations of the utmost importance,—yet aluminium has not come into use.

The reasons seem to be both physical and chemical. Besides a large area of depositing surface necessary, heating the solutions to 100° F. would seem to be compulsory, if excessive waste of aluminium and formation of a choking volume of aluminium hydrate are to be avoided. Again, if any lime be contained in the ore, or added for neutralising purposes, a thick deposit occurs on the aluminium, preventing access of the gold ; in hard waters, addition of NaHO is necessary (Dr. Gaze).

Numerous experiments were made with aluminium foil at Lucknow, and with every precaution as to current, temperature, and absence of lime or caustic alkalies ; the result was always the same, viz. very satisfactory precipitation for a short time, and then a heavy formation of gelatinous hydrate containing much iron, with great waste of cathode and interference with the operations.

Apparently Alfred James has experienced similar drawbacks in his investigations.*

*Stannous Chloride.*—The extreme delicacy of this reagent as a test for gold suggested it as a precipitant from cyanide solutions. But several circumstances combine to preclude it. In the first place, it is found that, even when presence of the stannic salt is avoided by keeping metallic tin in the solution,

* Trans. Inst. M. & M., vi. 225 (1898).

with weak gold liquors the precipitate does not assume a ponderable form, but rather constitutes a heavy cloudiness, and it is a matter of extreme difficulty to collect such a deposit. Again, it must be used in presence of an acid, which necessitates destruction of any KCy remaining in the liquor, so that the waste and cost would be great unless the auro-cyanide solutions were so attenuated in KCy contents that there would be a risk of the gold ceasing to be fully held in solution.

*Cyano-amalgamation.*—The precipitation of gold from its cyanide solution in mercury, or on amalgamated plates, or as an amalgam with some base metal, either with or without the aid of an electric current, has always been attractive because of the simplicity of the clean-up.

Probably the first process to be installed in practice was Molloy's, being a modification of his so-called hydrogen amalgamator, in which a shallow mercury bath forms the cathode, and has in its centre a porous cell containing potassium carbonate solution. A galvanic current is made to decompose the potassium carbonate, the potassium uniting with the mercury, forming potassium amalgam, which, in presence of aqueous solution of auro-potassic cyanide, decomposes the water, liberating hydrogen, whereupon the gold is reduced from the auro-potassic cyanide and combines with the mercury, while potassium cyanide is regenerated. Theoretically it looks perfect, but it very soon went out of use, apparently on account of the extreme inconvenience and costliness of the cathode, tons of mercury being needed in an installation of any size.

Hannay's method was much the same in principle, but the mode consisted of an annular band of powdered graphite and rosin compressed into a solid mass, and fixed several inches above the mercury bath. On 10-t. tests with pyritic and even telluric ores high extractions were got, but it has nowhere been adopted.

Another form is the Pelatan-Clerici process, in which the mercury bath has a copper plate lying at the bottom, the anode being an agitator suspended in the bath, and salt being added

to lessen the electrical resistance of the solution. Works (50 t.) were erected for this process at DeLamar (Idaho) in 1897, (35-t.) at the Republic mine (Washington) in 1898, and at one or two other places; but judging from newspaper reports, every one has been or is to be converted to some other method. A difficulty would seem to be that, under the influence of the current, the gold does not merely adhere to the surface of the copper plate as an amalgam, but permeates it, and therefore greatly hinders instead of facilitating the clean-up.

At the DeLamar mill, the ore treated assays about 12·15 dwt. gold and 3·02 dwt. silver, and is milled through 30-mesh screens in a Huntington mill and 24-mesh in a battery. It will only yield about ·25 to ·50 % concentrates, which are partially oxidised iron, copper, lead, and zinc sulphides. Pan amalgamation (raw) was previously adopted, and returned 40 to 50 % of the value.

The Pelatan-Clerici plant is outlined in Fig. 175.* The wet-crushed pulp is fed by a chain conveyor into 5 circular wooden agitator vats *a*, 8 ft. deep and 8 ft. 6 in. diam., using as little water as possible, generally about 1 to 1, which makes a thick batter. It is kept in slow motion to prevent settling by an ordinary agitator shaft and four arms (suspended from above, but not piercing the tank from below) until the agitator is filled to a mark previously established as giving the proper amount for a charge. Then the flowing pulp is turned into another agitator. When a precipitating vat is ready, it is charged by opening a 4-in. valve *b*, allowing the agitator charge to flow through a trough *c* to the precipitating vat *d*.

This latter is built of wood, 8 ft. 8 in. diam. and 4 ft. deep, with a copper plate covering the entire bottom, and a cemented rim to prevent pulp, mercury and amalgam from getting beneath it. Motion is given to the pulp by a hanging vertical shaft driven by spur gear *e*, terminating in 4 arms carrying wooden pegs *f*. On the lower part of each arm is a sheet of wrought iron, 4 ft. long, 14 in. wide, and ½ in. thick, forming the anode plate, by which an electric current from a dynamo

* D. B. Huntley, En. & Min. Jl., Aug. 7, 1897.

is passed through the liquid to the copper plate and mercury bath beneath, this last consisting of about 600 lb. and making a layer $\frac{1}{2}$ in. deep. A 2-in. aperture in the copper plate, leading to a 2-in. pipe with valve, is stopped with a rubber plug during operations.

When the precipitator is charged, which occupies only a few

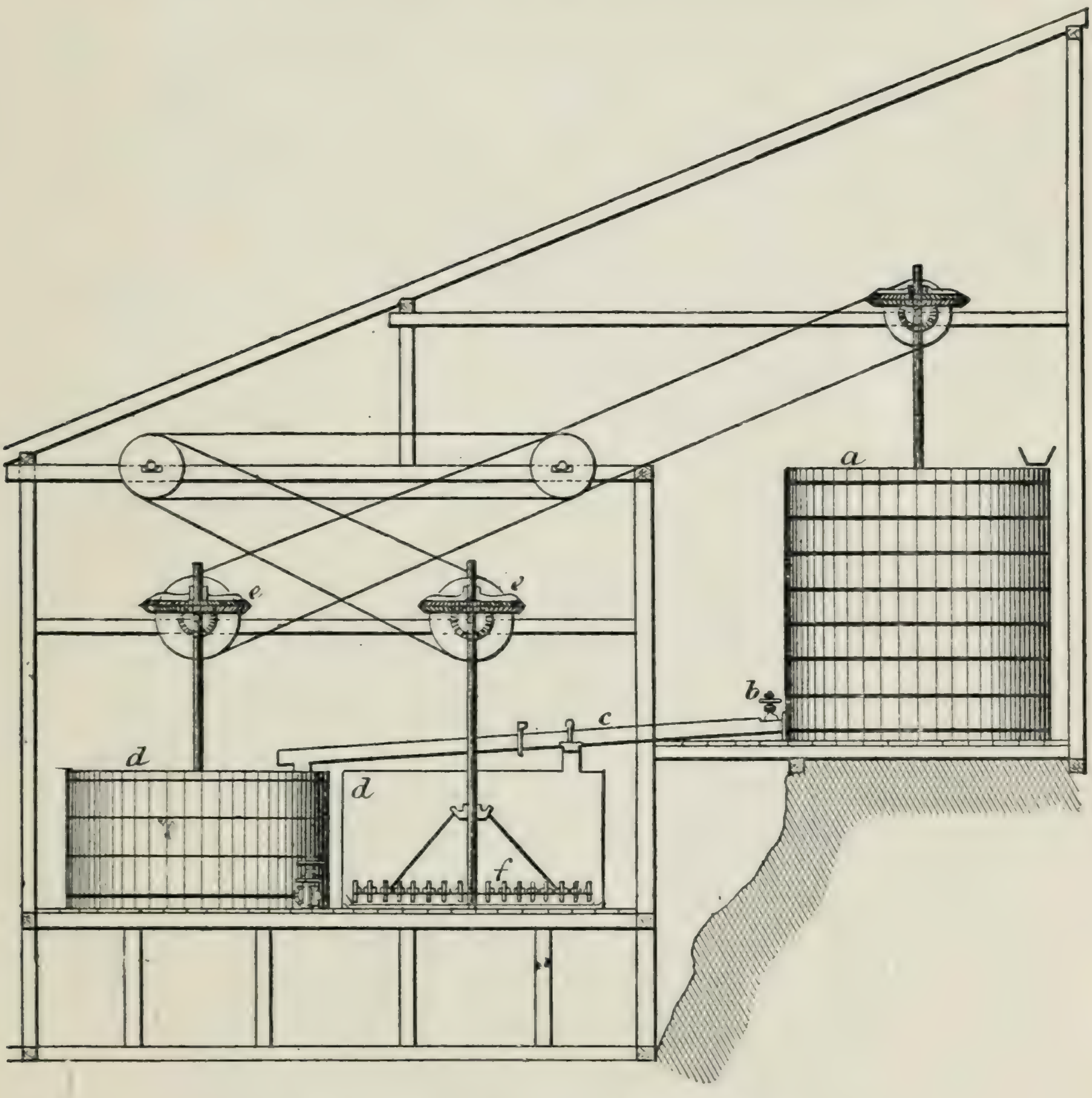



FIG. 175.—PELATAN-CLERICI PLANT.

minutes, for every ton of dry ore are added $2\frac{1}{2}$ to 3 lb. KCy and 6 lb. salt. Agitation at about 20 rev. a minute is maintained for $11\frac{1}{2}$ hours, and a charge being estimated at $2\frac{1}{2}$ t., the daily capacity of each precipitator is 5 t. At the end of $11\frac{1}{2}$ hours, the reactions are complete, and the pulp is allowed to discharge itself by opening a 4-in. gate in the side. Then

water is admitted, and men with brooms sweep out the residual sand, water, mercury and thin amalgam into a launder leading to the clean-up pan.

Practical results seem to be that fine grinding and constant motion in the precipitator are essential to avoid a layer of heavy sand forming on the mercury and preventing contact of valuable particles. Water consumption is small if the pulp be kept as thick as the process demands. Mercury losses range between 2½ and 8 oz. per ton. Extractions on 40-mesh pulp were 84 to 88 % ; on 30-mesh, 80 %, according to assay returns ; but the first month's actual clean-up was only 70·1 %, and the second 75·91 %.

At the Republic mill, it was found that 20 hours' agitation was necessary instead of 11½, and only 85 % extraction was obtained on 4- and 5-oz. ore and but 70 % on 1½-oz. ore, notwithstanding it was crushed to 120 mesh. Though this was a marked advance over the 40 % by simple amalgamation, it was not considered satisfactory, and seems to have been discontinued.

Others have advocated amalgamated copper plates, notably Andreoli and Dr. Keith, the latter using an E.M.F. of ½ volt and a current density of ·06 ampere per sq. ft. of cathode. The anode is not allowed to dip into the cyanide solution, but is placed in a porous cell and surrounded with a solution of an alkaline salt. It was reported that Keith's process was to be adopted for arsenical tailings at Pestarena in 1897, but no results have been published. The process also embraces the use of ·025 % cyanide of mercury in conjunction with ·05 % KCy.

The so-called "aurex sluice" is another development of the same idea, with modifications which aim at rendering the process continuous instead of intermittent, and minimising the expenditure of electrical energy by bringing the precious metals to the electric current by the exercise of their own gravity. The apparatus is a trough, upon the floor of which are laid curved wooden blocks closely abutting. Upon these are placed amalgamated copper plates curved to fit them. The plates are slightly smaller than the blocks, to admit of easy

removal and replacement, therefore they do not touch each other, the electrical connection being made by means of mercury poured into the intervening spaces. This also ensures connection with the negative electrodes, the copper plates being the cathodes of the cell. Above each plate is placed its anode, which is a lead plate curved approximately like the cathode, but so differing that, when the sluice is given its proper grade, the lower edge of the anode is nearer the cathode than the upper one. As this lower edge dips into the valley formed by the curves of two adjacent cathodes, and the stream of solution carrying the ore flows through the space between the anodes and cathodes, the effect of this contraction is to produce a slight increase of pressure in the flow of solution, just sufficient to prevent the heavier ore particles from settling in the valleys. This increase in pressure can be regulated both by the grade of the sluice and by causing the lower edge of the anode to approach or to recede from the cathode, and should be just sufficient to keep the ore in motion without causing any scouring action upon the cathodes, so that ore of any of the usual mill grades of fineness can be handled. By this form of anode, the current of electricity, which always takes the path of least resistance, may be expected to exert its greatest effect at the lower end of the anode, just where every particle of the ore is brought into most intimate contact with the nascent cyanogen liberated by the action of the current.

From a number of tests made by T. M. Chatard and C. Whitehead,* it would seem that, operating on ore containing 2·68 oz. gold per ton, the following results were got:—

| Solution. | Current. | Tailings. | Extraction. |
|---|---|---|---|
| ·15 % KCy | None | ·90 oz. gold | 66·42 % |
| ·10 ,, | 3·5 volts, ·64 am. | ·45 ,, | 83·21 ,, |
| ·276 ,, | 2·8 ,, ·57 ,, | ·27 ,, | 89·92 ,, |

* En. & Min. Jl., Feb. 3, 1900.

These figures in themselves are not particularly convincing, as admittedly over $\frac{1}{4}$ oz. gold per ton is not recovered. The authors state that they found a current of ·44 ampere not to be destructive to cyanide, which is widely at variance with the results obtained by other investigators. They confess that "experience has shown the need of substantial modifications in the design" of the plant.

Recently, Dr. T. K. Rose * made a series of experiments on electrical precipitation of gold from cyanide solutions on amalgamated copper plates, from which he early concluded that current-density is the main factor. Satisfactory precipitation can only be obtained when a low current-density is employed. With ·01 ampere or less per sq. ft. of cathode, gold is precipitated and amalgamated simultaneously, the plates keeping in good condition during 24 hours' use, while $\frac{1}{2}$ oz. gold per sq. yd. of plate is deposited on them. With currents of greater density, part of the gold is deposited as a non-adherent black powder easily rubbed off, but not easily detachable by a jet of water, and not easily amalgamated. As the density of the current is increased, the percentage of non-amalgamable gold thrown down rises, until at ·30 ampere per sq. ft. about half the precipitated gold is not amalgamated. Moreover, decomposition of cyanide is greater when amalgamated copper electrodes are used than if lead and iron plates are substituted for them. On the other hand, more gold appears to be deposited on copper than on lead, especially when rich solutions are being treated.

The conclusions to be drawn from the experiments do not appear to lend any countenance to the view that mercury or amalgamated copper plates will ever be satisfactory cathodes for electrolysis of cyanide solutions. A current must be of small density in order to deposit gold on mercury in a form that admits of easy recovery, and the high cost of mercury prohibits its use. With amalgamated copper plates, the current-density must apparently be even smaller, not exceeding, say, ·015 ampere per sq. ft., in order that the gold may be deposited and retained, and the dimensions of the cathodes

* Trans. Inst. M. & M., viii. 369 (1900).

must, therefore, be even greater than if mercury is used, besides which the cyanide solution suffers loss by dissolving copper from the cathode. Against all these disadvantages, there appears to be no real advantage. Thus scientific investigation endorses the condemnation pronounced by the practical man.

Sodium amalgam has been tried, and, according to Dr. Simon, on the small scale in the laboratory, it "worked in a perfectly ideal fashion ; but, as soon as attempted on a large scale, complicated reactions take place, which prove it to be an utter failure."

A measure of success seems, however, to have attended the use of copper and copper-zinc amalgams, at least on a property in Nicaragua, where it was introduced by Gilmour and Young,* and applied to several tons of ore carrying so much kaolin that the slimes amount to 70 %. By dry crushing to 30 mesh and agitating the pulp with mercury and KCy solution, an extraction of 90 % of the gold and 70 % of the silver has been won from slimes assaying 1½ oz. gold and 4½ oz. silver, by 6 to 7 days' treatment, and at a cost of 1¾ lb. KCy per ton. The silver occurs as black sulphide ; pyrites is present in very small amount, with a minute quantity of copper. All decantation and filtration are avoided, the gold being precipitated in the agitators.

There are 4 Boss pans, 5 ft. diam., and 2 settlers, 7 ft. 6 in. diam. The dry-crushed ore is charged into a pan in 2-t. lots, with 100 gal. water, so as to form a very thick pulp. Some 2 to 6 bottles of mercury are added, until the globules can be seen circulating in the pulp, and then the required amount of cyanide. The speed must not fall below 68 rev. a minute, and the mullers must be set as low as possible without grinding. After running this for about 2 hours, 10 lb. mixed zinc and copper amalgam are added, and the pan is run for 4 hours longer. By this time, precipitation of gold from the solution is stated to be very complete, only 10 gr. of gold per ton remaining in solution, and this has even been reduced to 1 gr. per ton of ore. The pulp and solution are then dis-

* Trans. Inst. M. & M., vii. 63 (1899).

charged into a settler, and the mercury is recovered as in the ordinary pan process.

The quantities and cost of chemicals used are given as follows :—

| | lb. | | s. | d. | | s. | d. |
|---|---|---|---|---|---|---|---|
| Cyanide | 1·67 | at | 1 | 3 | per lb. | 2 | 1·05 |
| Caustic soda | 0·63 | ,, | 0 | 1 | ,, | 0 | 0·65 |
| Copper sulphate | 1·16 | ,, | 0 | 1¾ | ,, | 0 | 2·03 |
| Cast-iron turnings | 0·25 | ,, | 0 | 0½ | ,, | 0 | 0·12 |
| Zinc | 0·39 | ,, | 0 | 3 | ,, | 0 | 1·17 |
| Mercury | 0·40 | ,, | 1 | 9 | ,, | 0 | 7·81 |
| Total cost per ton | | | | | | 3*s.* | 0·8*d.* |

these figures applying to rich ore carrying about 3 oz. gold per ton. Labour is estimated at 3*d.* per ton, and 24 h.p. is required for 50 t. per diem.

Copper sulphate and cast-iron turnings are used to make the copper amalgam in a small separate pan, a convenient charge being 100 lb. copper sulphate, 30 lb. iron filings, and 6 bottles mercury ; the reaction is complete in 2 to 4 hours. The zinc amalgam is very easily made by pouring mercury into molten zinc ; it is advisable to keep it under water, to prevent rapid decomposition.

In practical working it is found that zinc amalgam alone presents great physical difficulties in the settlers, as it is liable to crumble into powder and flow off with the water, and will even blow up into light spongy masses and oxidise, giving off great heat. The addition of copper amalgam prevents all this, and gives a stable amalgam to work with. Copper amalgam alone gives as good precipitation as zinc under certain conditions, particularly with solutions weak in KCy.

The amalgam is retained in use until it becomes well charged with gold, and in this way a much purer bullion is got than would otherwise be the case, the repeated action of the cyanide being to dissolve away the zinc and some of the copper, leaving in the end a fairly good amalgam. This becomes less effective as a precipitant in proportion to the diminished amount of zinc and copper present, and necessitates the use of larger quantities.

Cost for power depends so much on price of fuel and efficiency of boilers and machinery, that it is a matter depending much on locality. In Nicaragua, firewood is employed, costing 4*s.* 6*d.* per cord, 4 cords being equal to 1 t. of ordinary coal, say 18*s.*; 3 lb. of coal per h.p. per hour is a fair allowance, so that in this instance power is a small item in the total cost.

It would seem that the method may have a limited sphere of usefulness.

*Copper Sulphide.*—This has been suggested as a precipitant by Prof. De Wilde, and in the laboratory it works splendidly, giving almost analytically perfect returns. But Bettel * has shown that it is not practical for ordinary working conditions. In a plant treating 600 t. slimes a month, if all the KCy is to be thrown down as cuprous cyanide, there will be 1800 t. of ·01 % KCy solution assaying 20 gr. per ton, which will give a precipitate of 522½ lb. dry cuprous cyanide, and about 9 lb. mixed double cyanides (silver-copper and gold-copper), besides 180 lb. cupric ferro-cyanide, or a total of 711½ lb., from which to recover 75 oz. gold and 7½ oz. silver. The additional cost of handling is computed at 7*s.* 5*d.* a ton. But Bettel thinks it might be applicable to cupriferous ores, by using $SO_2$ to neutralise alkalinity of solution and reduce cupric to cuprous salts; and Williams entertains a similar opinion of its chances of economical adoption where much soluble copper is in the ore and where all free KCy has been eliminated from solutions.

**Clean-up.**—Not only does the nature of the clean-up differ with the precipitant used, but even with the same precipitation method variations occur in the arrangements of different works.

*Zinc Shavings.*—The clean-up of zinc-deposited bullion involves mechanical operations to efficiently separate the gold "mud" [often called "slime," which term is here reserved throughout for application to fine ore pulp] from the remnant of undecomposed zinc, and the drying of that mud, with or

* Jl. Chem. & Met. Soc. S. Africa, ii. 48 (Feb. 1899).

without the interposition of chemical treatment to remove in solution as much as possible of the zinc which has escaped mechanical recovery.

According to Eissler,* the trays holding the zinc shavings are lifted out from the compartments and pulsated up and down in the solution, so that the fine particles of gold mud and zinc fall through the sieve and settle in the bottom of the box ; but before starting this operation, it is advisable to run into the box a sufficient amount of clean water to remove the cyanide solution, as it is injurious to white workmen, though Kaffirs put their arms and hands into it without being affected. The zinc shavings are taken out of the tray, which is placed on a rack above the box, so as to allow the solution to drain back into the box. They are also rubbed in the water to remove gold adhering to them, and the tray is turned over and brushed down with a like object. The zinc shavings get very hot, on account of oxidation,—(it will be noticed that steam rises from them)—therefore they should be exposed as little as possible to the air. The solution or water in the zinc boxes is pumped into settling tanks, where it is allowed to stand for two weeks, so as to give the extremely fine particles of gold held in suspension time to settle. In pumping out the precipitation boxes, great care is taken not to disturb the gold-zinc mud in the bottom ; the water is pumped out to within 2 in. of the mud, and the rubber suction hose is then moved into the second compartment, and so on. The mud is pushed back to one corner, and the supernatant liquor is allowed to stand for a while, after which it is pumped into the settling tanks. The mud is then scooped into enamelled iron buckets, and discharged on to a fine sieve, say of 900 mesh, and washed and rubbed into the gold clean-up tank. After the water is settled here, it is siphoned or pumped off, and the precipitate is drawn off through plug holes on to a calico or linen filter, or into a filter press. The zinc shavings, some of which have quite a brown coating of gold, which cannot be removed even by rubbing, are again returned to the precipitation boxes, and fresh zinc is put on the top. The gold

* Trans. Inst. M. & M., iii. 66 (1895).

which sticks to the zinc will be recovered in the next clean-up. After the gold mud is sufficiently dry to be handled with a scoop, it is dried on an iron plate or in iron pots, and is then ready for melting, or for the intermediate processes of roasting or leaching, when such are adopted.

The object of roasting is to oxidise as much as possible of the zinc fragments which have contaminated the gold mud, though a portion of this oxide will be reduced in melting and re-enter the bullion. Feldtmann recommends addition of 3 to 10 % of nitre, as a strong solution, before drying the gold mud, "so that it gets equally mixed with the whole mass. In the subsequent roasting, the nitre not only assists by yielding up oxygen to the zinc, but to some extent also appears to flux the zinc oxide, forming zincate of potash, which is not so readily reduced as zinc oxide. In case the precipitate is very sandy—owing to tailings coming through the filters—nitre roasting is not so successful, as it tends to cake. By addition of nitre, the tendency of the precipitate to dust on stirring up in the roasting furnace is minimised, the amount of flux required in smelting is reduced, and the resulting bullion is better. In roasting the precipitate, care should be taken not to raise the temperature much above a dull-red heat (to avoid partially fusing it to a pasty mass), and not to stir too violently, especially at commencement of roast."

The method mentioned by Alfred James * differs in some details, all the operations being performed in the extractors (precipitators), so that risk of mechanical loss is reduced; further, no acid treatment or roasting with nitre is involved, as not more than 1 % zinc is reckoned to remain in the mud. As before, fresh water is allowed to flow through the extractors until the effluent is no longer decidedly alkaline. The zinc shavings in the upper compartment are rubbed between the fingers to facilitate disentanglement of the gold mud, which settles in the bottom of the compartment. The tray is withdrawn, the supernatant liquor is baled into the next lower compartment, and the gold mud is allowed to flow into the launder, and down to a 40-mesh sieve about 2 ft. sq. This

* Trans. Inst. M. & M., iii. 403 (1895).

sieve is suspended over the settler, and has attached to it a twill bag, with a 3-in. slit just under the sieve. The fine gold mud is carried through the 40-mesh sieve, whilst the disintegrated zinc remains behind. This is scrubbed with a piece of flat rubber to still further disengage the mud, and the retained zinc is put back on to the tray, which has meanwhile been cleaned and replaced in the thoroughly washed compartment. The second compartment is treated similarly, and the zinc, after being freed from gold mud, is moved up into the first compartment until this contains its normal amount of shavings. This operation is continued similarly with the other compartments. Thoughout the process the cleaned zinc is moved up, so that fresh shavings are put into the lower compartment only. The filter bag under the sieve becomes full of gold mud, and the liquid overflows through the slit into the settler, carrying a portion of the mud with it. As soon as the extractors have been cleaned up, the filter bag is squeezed to press out any liquid, and the contents are dumped out on to a heated iron plate, with turned-up edges, on which the mud is dried and mixed with fluxes in readiness for melting down. Alum is added to settle the contents of the settlers, and the supernatant liquid when clear, is siphoned off and tested for gold; the mud is collected from the pyramidal bottom, filtered and dried. Sometimes a false bottom is fitted into the settlers, and the liquid is drawn off by vacuum apparatus, which leaves the mud in a cake convenient for handling.

The fineness of the sieve through which the mud passes makes a great difference in the purity of the bullion obtained. "When 30-mesh only is used, a considerable amount of zinc passes through, and the resulting bullion is very base. With a 40-mesh sieve and well-turned zinc, bullion has been obtained above 960 fine—gold and silver—without any acid treatment, roasting, or special fluxes." (A. James.) This condition would seem to be encountered only where silver forms a very large part of the bullion.

Another modification, proposed originally by Prof. A. K. Huntington in the early days of the cyanide process, and since

come into favour, is a cylindrical sieve revolving under water for separation of the unused zinc.

The Cassel Gold Extracting Co. employ filter-cloths, as described by their chief chemist, C. J. Ellis.* At the exit of the launders from the extractor boxes, are placed small vats, with false bottoms covered with cloth, and connected with a chamber in which a vacuum can be produced. After the usual water wash to displace KCy solution in the boxes, the valves or plugs of the various compartments are opened one by one, and the gold mud is sluiced out into filters. The larger portions of zinc thread are gathered up on a sieve, and a hose is used to swill out all remaining deposit. The mud adhering to the zinc thread on the sieve is detached by vigorous stirring in water and added to the main portion in the filters; the zinc itself is returned to the boxes, the necessary fresh zinc is added, and the flow of solution is restarted. Practically the whole of the water from the gold mud in the filters is then separated by filtration accelerated by suction. The moist mud is dried, and, as it always contains a considerable quantity of mercury, is retorted. Treatment with acid after retorting, to remove base metals, has been discontinued. The retorted mud is roasted, in some cases having been previously drenched with a little strong solution of nitre.

Acid treatment of gold-zinc mud was introduced by T. H. Leggett, both at the Standard Consolidated mill, Bodie (California) and at the Treasury mill (Transvaal), and is thus described by him.† The mud from the extractor boxes is placed in the wooden tank *a*, Fig. 176, where also the zinc shavings are washed through a coarse screen. The cyanide solution brought with the mud and shavings from the boxes is then pumped out by means of rubber hose *b* attached to pump suction *c* and is forced through filter-press *d*, leaving the mud in the bottom of tank in a thick condition. A bucket of concentrated sulphuric acid is then poured upon the mud, dissolving the zinc. To prevent the fumes poisoning the air of the room, a hood *e* lowered on to the tank leads them through the roof. When the acid is spent, the hood is raised

* Trans. Inst. M. & M., iv. 257 (1896).

† *Ibid.*, v. 147 (1897).

and another bucketful is poured in, and so on until all the zinc is dissolved, when the contents of the tank should be about neutral. The boiler *f* is now filled with cold water through pipe *g* and brought to boiling by steam entering from below at *h*. This boiling water is run into the tank on to the gold mud (by means of rubber hose *i*) until the tank is filled, the mud meanwhile being agitated with a wooden stirrer

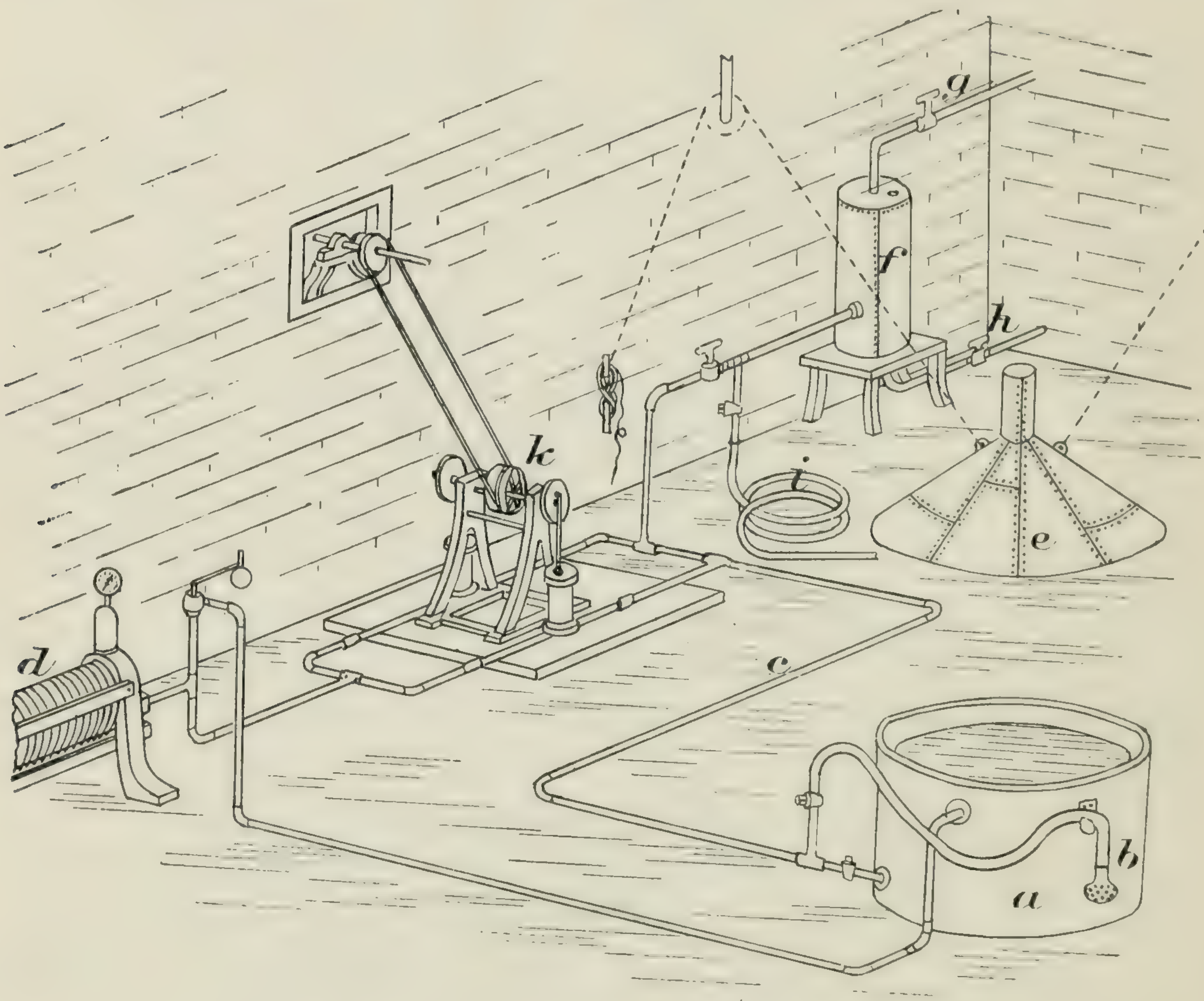



FIG. 176.—ACID PLANT FOR GOLD MUD.

The pumps *k* draw off this hot slimy solution and force it through the filter-press *d*, the solution being stirred all the time, and more hot water being added as required, until the tub is empty. More hot water is then pumped through the filter to further wash the mud cakes, and to remove all acidity and zinc sulphate. Air may now be pumped through to dry the filtrates, after which the press is unscrewed and the mud cakes are removed. These are dried in a closed muffle and without stirring, so that absolutely no dust is made. The

dried slimes are melted into bars in the usual way. In a bi-monthly clean-up, 108 lb. (1575 oz.) dried gold mud uses 234 lb. sulphuric acid, or about 150 lb. per 1000 oz. dry precipitate, and the consumption of hot water is about 400 gal. per 1000 oz. Bullion of 843 fineness (750 gold, 93 silver) is obtained, and the effluent liquors from the filter-press assay about 2 gr. per ton. There is an important advantage in merely drying the gold-mud cakes without stirring, and the application of hot-water washes at 60 to 80 lb. per sq. in. pressure thoroughly eliminates the zinc sulphate, and reduces the total loss to below ·1 %, while giving higher bullion than by nitre roasting.

Filter-presses are specially made for this work, both Johnson's and Dehne's being in use. They may be provided with plates of the best pinewood, frame and filtering surface being cut in one piece, and the sections held together by means of rods running horizontally through at top and bottom, which do not come into contact with the liquors to any extent; but more often the plates are of iron, coated with black bituminous enamel. The valves, and channels through iron head-pieces, are lined with hard lead, and a tap is provided for each chamber. Approximate prices for presses with 30-in. sq. plates are from 55*l.* for 12 chambers to 86*l.* for 30.

At the Standard plant, filtering in a Johnson press, the out-going solution does not show more than the faintest film of sediment, after standing 24 hours, while the average of many tests made by evaporation of this falls below 2½*d.* per ton of solution; in this case, it may be assumed as approximately correct that 1000 oz. of precipitate are washed by something under 2½ t. of water, implying a loss of 6¼*d.*, or less, per 1000 oz.*

But it was subsequently found that the filter-press method became inapplicable,† because of the fine state of division of the precipitate, the pressure frequently running up to 100 lb. on the gauge, and the press being found to contain only a thin coating of gold mud on the cloths. The press has there-

* R. G. Brown, Trans. Inst. M. & M., iv. 250 (1896).

† *Ibid.*, v. 150 (1897).

fore been discarded, and a vacuum filter-box (Fig. 177) used in place of it. This is a wooden box *a* of $2\frac{1}{2}$-in. wood, carefully made, and sustained against collapsing pressure with suitable struts *b* and bolts *c*. It is 30 in. sq. by 48 in. deep,

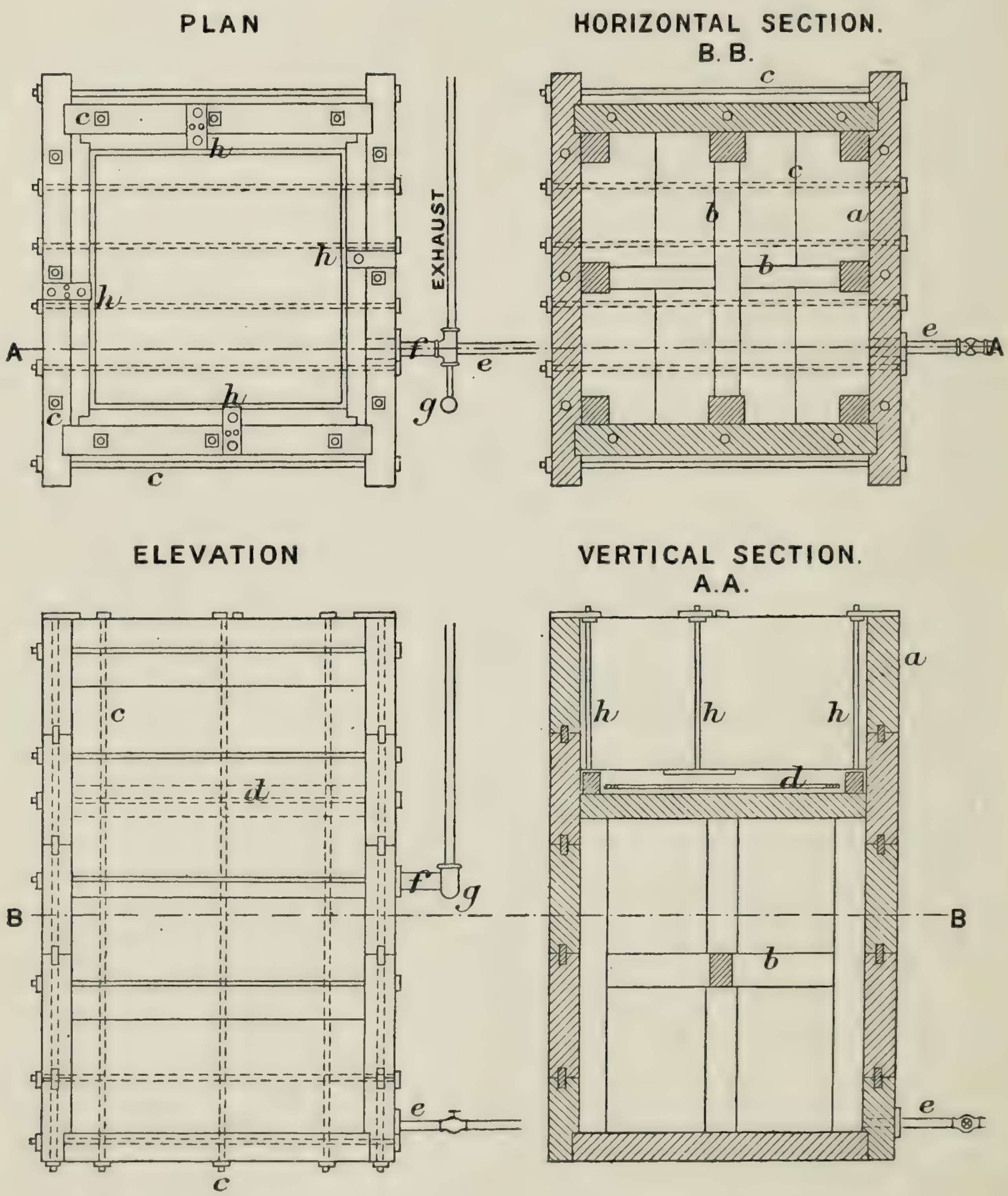



FIG. 177.—VACUUM FILTER-BOX.

with a filter-bed *d* placed 14 in. from the top. At the bottom is a drain pipe *e* with valve, and close below the filter is another pipe *f*, connected with steam aspirator *g*, which produces the vacuum. The filter-bed *d* is of wood, perforated at $1\frac{1}{2}$-in. intervals by $\frac{1}{16}$-in. holes, and covered with fine (30-mesh) wire

screening, and with double mill blanket for actual filtering medium, the blanket frame being held down by bolts *h*. The acid treatment is effected in a wooden tank 6 ft. diam. and 2 ft. deep, and a settling tank of 12 t. (2900 gal.) capacity is provided.

The drying appliances at the same plant consist of a rectangular brick furnace, with iron top, an iron pan, and a hood to cover the same, connected by some half-dozen lengths of 7-in. stove-pipe with the outside. During a special test made in drying about 4400 oz. of precipitate, there was collected from this flue-dust a value of 14*s*. 8*d*. While this test was in progress, the precaution was taken to cover the end of the pipe with soft, closely woven fabric ; but, although this was blackened by soot from the furnace, when burned, the ashes showed but a trace of the precious metals. The loss in value of this lot of precipitate was ·03 %. Several pounds of mercury were recovered from this flue during the year.

The whole cycle of operations is as follows :—The zinc cleanings from the precipitation boxes are thrown on a sieve of 50 meshes to the linear inch over the filter-box, and the fine mud is washed through on to the filter ; the water is drawn through the filter by means of the aspirator, and runs to waste, being perfectly clear, and containing practically nothing. The zinc from the sieve is transferred to the acid tank, and treated with acid in successive charges ; the fine precipitate is removed from the filter, treated separately in a tub, and returned to the rest after acid treatment and decantation. Hot water (about 200 gal.) is then run on, the whole is settled and decanted through a siphon into the settling tank, and the residues are washed three times, or until there is no acid taste in the liquid, all wash-water going to the settling tank. The whole mass of gold mud is then drawn from the bottom of the treatment tank on to the filter, and the water is sucked out. If the bed becomes too thick, the material can at any time be scraped up on to one end of the filter, and the filtration be proceeded with. Air is then sucked through, when the precipitate is compact enough to be removed in bricks. For drying, the cakes are scored across with a knife,

but there is no stirring, and consequently no dusting. The product as it comes to the melting room is in lumps, and with scarcely any dust, and so inappreciable is the loss from this cause that the total value of material recovered from the dust flue and stack of the melting furnace in 1896, from the melting of 40,000 oz. of precipitate, was 10*l*. After a week's settling, the water is decanted from the settling tank, and fresh washings are run on. About once in 3 months the tank is cleaned out, and the recovered precipitate is melted. This amounts to 2 to 2½ % of the total, and the decanted liquor carries no more in value than the filtered liquor, possibly 2½*d*. per ton.

Both Brown and Leggett emphasise the importance of thorough washing out of zinc sulphate. With the presence of this impurity, not only are the losses by volatilisation increased, but also the cost of the melting, which may be even doubled or trebled.

At the Princess Works (Transvaal) * the zinc-box mud is separated from the solution by aid of a vacuum filter. A water-wash is passed through until the mud is free from cyanide. The gross weight of the mud, including moisture, is then taken, by weighing the buckets of moist mud during transference to a large sheet-iron tray placed alongside the acid tank, to determine the amount of sulphuric acid necessary to destroy the zinc. Having found the approximate weight of mud to be treated, sufficient water is run into the acid-vat to form, on the addition of the acid, a 10 % solution : 1 lb. of acid for every lb. of moist mud gives good results. This would be equivalent to about 1⅓ lb. acid to the lb. of dry mud. The acid is then added and the vat is closed down tightly. The stirring apparatus is kept continually moving during the time of feeding-in the mud, which is done gradually in the condition in which it was taken from the filter-vat. It is beneficial to keep up a continual stirring for at least ½ hour after the action has apparently ceased. After all the mud is in the acid, a jet of water is turned into the hopper to wash down any adherent mud, and everything that has been used in the cleaning-up of the boxes, etc., is well washed in the same jet. The stirring

* E. H. Johnson, Jl. Chem. & Met. Soc. S. Africa, June 19, 1897.

apparatus is then detached and well washed in the vat during removal. The vat is then filled with water and allowed to settle. Working with dilute acid, and not heating, a perfect settlement takes place within an hour. When heating with a steam jet, settlement is much more difficult. The washing is done by siphoning off the clear liquor, and filling the vat repeatedly with water, until the solution is neutral to litmus paper—usually 4 or 5 washings. It is well stirred at each refilling by means of a long wooden paddle, a rotary motion being given to the water. This causes the mud to collect in the middle of the vat, and reduces the risk of loss during siphoning—the siphon being let down at the side. A sample of the washings taken continually during siphoning off showed, on assay of a large sample, 13 gr. of gold per ton of solution.

The drying of the resultant gold mud is conducted on an open drying hearth in large cast-iron enamelled dishes. The cakes are subsequently broken up and transferred to small sheet-iron trays in thin layers, and subjected to an increased heat. When cool, the mud is ground, fluxed, and transferred to the crucible. It fuses quietly and with but little fume, and normally yields 50 to 60 % of its weight as bullion. The average fineness of the bullion is about 820. The slag after panning out the prills (of which there is very little) assays 23 oz. per ton, and one ton of slag has been accumulated in two years in producing 11,627 oz. of fine gold, which is equivalent to a little under 0·2 % of the total gold locked up in slags. The cost of reduction, including acid, is 6·7*d.* per fine oz., made up as follows, taking an actual "clean-up" as a basis: Dry weight of zinc-gold mud, 504 lb.; dry weight after acid treatment, 100 lb.

| | £ | *s.* | *d.* |
|---|---|---|---|
| 672 lb. acid at 4½*d.* per lb. | 12 | 12 | 0 |
| 66 lb. borax at 37*s.* 6*d.* per cwt. | 1 | 2 | 1 |
| 9 lb. carbonate of soda at 2¼*d.* per lb. | 0 | 1 | 8 |
| 9 lb. fluorspar at 4*d.* per lb. | 0 | 3 | 0 |
| 5 bags coke at 8*s.* 6*d.* per bag | 2 | 2 | 6 |
| 1 No. 60 crucible | 1 | 7 | 6 |
| Total | 17 | 8 | 9 |

Yield, 620 oz. fine gold, or 6·7*d.* per fine oz.

Roasting has been widely adopted in Africa, Australia, and New Zealand. In many cases the "furnace" is a very rudimentary arrangement, consisting of little but a cast-iron tray placed over a fireplace, and overhung by a sheet-iron hood to collect the fumes. These are given off copiously and must not be breathed. The heat reached should not exceed dull redness. Stirring frequently, but with great care to avoid dusting, is necessary to complete oxidation of the zinc.

A number of experiments made by Alfred James* indicate that the less the gold is handled, the less shortage it suffers, and that roasting with nitre is by far the most fertile source of loss, ranging from a minimum of ·23 % to a maximum of 2·57 %. Acid treatment yielded results varying with the kind used, nitric showing a greater loss than hydrochloric and sulphuric. With impure zinc in ordinary use, nitric acid gave the purest bullion ; but with lead-free zinc, dilute sulphuric gave both the purest bullion and the lowest acid loss. A comparison between melting direct and preliminary treatment with sulphuric acid gave results depending on the purity of the gold mud ; when this had been passed through a 40-mesh sieve to free it from coarse zinc, melting direct gave a fairly pure bullion, and at least as good results as with acid treatment ; but with scrap and coarse zinc included, the advantage lay with the latter method. The least total loss in 19 experiments was ·75 %, and this was obtained by treatment with sulphuric acid before melting. Lead-free zinc gave a total loss of only ·52 % with direct fusion of the sieved mud, the coarse zinc being treated with sulphuric acid and the residue added to the mud. Pure zinc invariably showed a smaller loss than ordinary zinc, averaging ·2 % advantage. Treatment with strong solutions of alkali and cyanide, with or without subsequent acid treatment, gave heavier losses. The average figures arrived at by James were :—

| | |
|---|---|
| Roasting, then melting, creates a loss of . . . . | ·23 % |
| Sulphuric acid, then melting . . . . . . . . . . | ·17 |
| Sulphuric acid, roasting, and melting . . . . . . | ·40 |
| Nitric acid, then melting . . . . . . . . . . . | ·29 |
| Melting only, with pure lead-free zinc . . . . . | ·43 |
| Melting only, with ordinary zinc . . . . . . . . | ·63 |

* Trans. Inst. M. & M., vi. 5 (1897).

His conclusions are that with either lead-free zinc or with ordinary zinc, direct melting gives the least loss of bullion, and that sulphuric acid treatment is the safest purifying means. Lead-free zinc obviates the need of any purification, and diminishes loss. Roasting causes heavy loss, and is less effective than acid in raising the grade of bullion. James gives preference to a detachable steel-lined filter-vat with vacuum connection over a filter-press for washing and drying the gold mud, as being cleaner, more easily handled, and delivering the gold mud in a single solid cake, thus avoiding the various plates, cloths, etc., of a press, in all of which some value is sure to be arrested.

At the works of the Mysore Co., India, the gold mud is retorted to free it from mercury, which is collected at the rate of 100 lb. a month, the cyanided tailings in this instance being derived from pan amalgamation. The mercury produces much floured and brittle zinc in the boxes. After retorting, the gold mud is mixed with 10 % nitre and roasted at bright red heat; then boiled with dilute (1 : 2) sulphuric acid, to dissolve out some of the copper (over 38 % of the bullion), reducing it to about 7 %.*

Similarly at mills working on the Comstock tailings, which have passed through the Washoe process of pan amalgamation, special provision has to be made for recovery of mercury, which forms quite an asset. The gold-zinc mud is dried and retorted in a "boat" inside an ordinary amalgam retort, and the vapour, after passing through the usual condenser, is directed through a hydraulic seal in a vacuum-chamber.

When lead is introduced by coating the zinc with plumbic acetate, the gold mud may contain a high percentage of this metal, and T. L. Carter mentions † an instance reaching 23 %.

*Zinc dust.*—At the Mercur mill, Utah, the gold mud is retained partially in the tanks and partially in the filter-presses through which the precipitate is washed, the tanks

* L. Pitblado, Jl. Soc. Chem. Ind., Feb. 28, 1898.

† Jl. Chem. & Met. Soc. S. Africa, Oct. 1898.

yielding 3 times the volume of mud but only about half the gold. The product is carried to the refinery, dried in shallow iron pans in a muffle furnace (where most of the zinc is converted into oxide), pulverised through ¼-in. screen, and sampled, averaging about 20 % gold. It is next treated in a dissolving tank with sulphuric and nitric acids, which remove remaining zinc, zinc oxide, arsenic and mercury, and bring the product up to 60 % gold, the other 40 % being mainly silicious matter. This is filter-pressed, dried in a muffle furnace, and crushed roughly to ½ in. or less, ready for melting. An excessive amount of handling is involved, and the whole system compares very unfavourably with that at Bodie (p. 643).

At Deloro the precipitate of gold mud, as drawn off from the cones, varies in gold richness from about 8 % to as high as 30 %. It is all first carefully dried in open enamelled iron pans, care being taken to prevent dusting and consequent loss. When dry, it is mixed and sampled for assay ; perfectly reliable samples can be taken by quartering down, owing to the even distribution of the gold throughout the mass. The assay of the mud therefore gives a good check on the extraction, as shown by liquor assays.

Of the usual methods of treating the precipitate for its gold-contents,—the acid process (which consists in dissolving out the remaining zinc with sulphuric acid, filtering and smelting), and the method of roasting the whole mass (to oxidise the base metals present, and subsequently fluxing and smelting)—neither was adopted : the first, chiefly on account of its high acid cost ; the second, partly on account of cost, partly owing to the difficulty of smelting the base oxides (whereby large quantities of bad slags very rich in gold are necessarily produced), and, finally, because of the serious gold loss which inevitably occurs with such a method. After careful consideration, it was decided to distil the zinc from the dried precipitate, leaving the gold behind in the retort.

To effect this, a Balbach tipping-furnace capable of taking a No. 3 Dixon retort was employed. This furnace (Figs. 178–180) is built of fire-brick *a* in a heavy cast-iron frame *b*, the whole mounted on trunnions *c*, so arranged that by

means of a geared hand-wheel *d* the entire furnace with its contents can be tipped. The retort *e* rests at an angle of about 45° on a fire-brick arch *f* built into the furnace just above the fire-bars *g*. Its neck *h* just protrudes beyond the front wall of the furnace.

When the charge was introduced dry, and distillation was

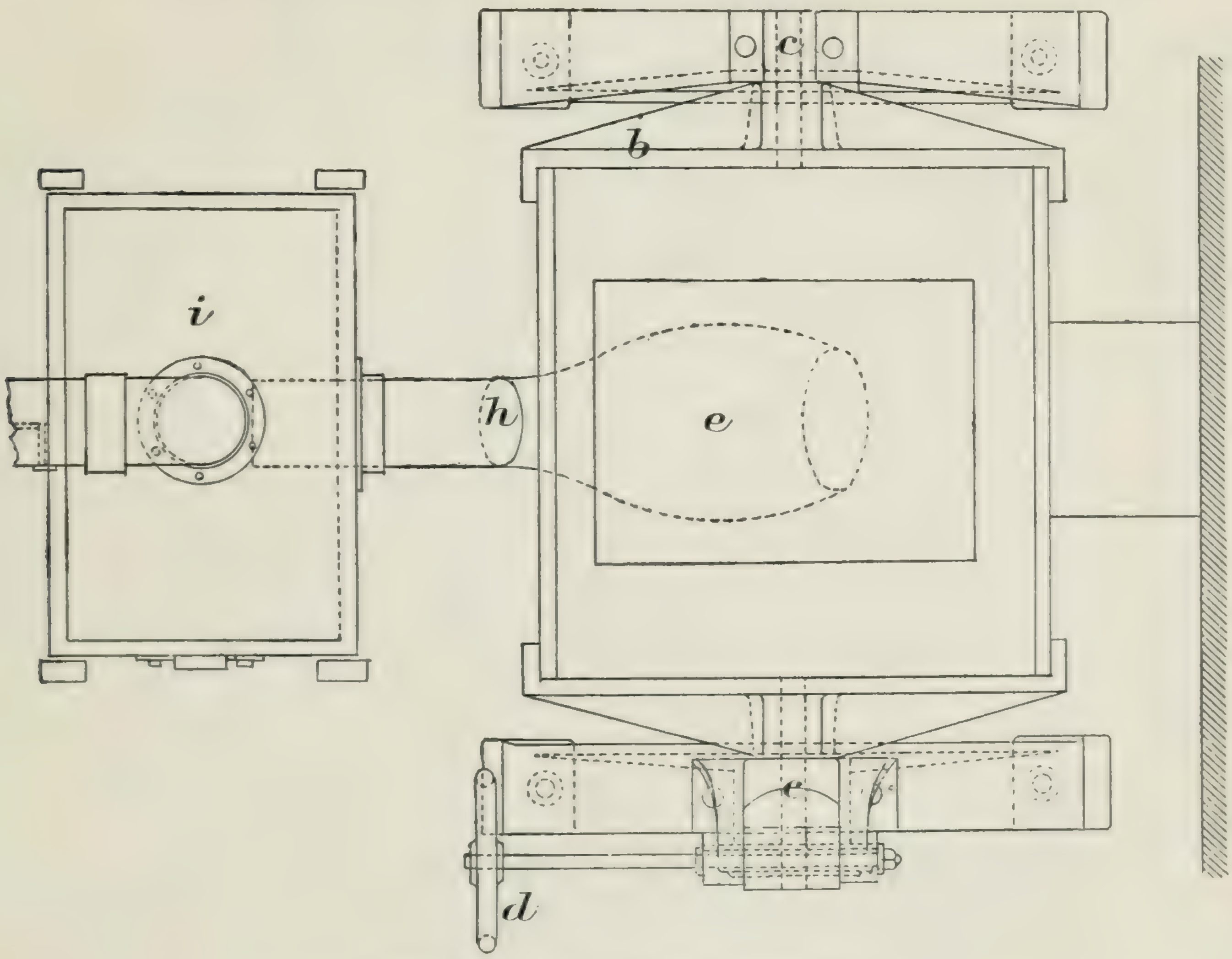



FIG. 178.—BALBACH FURNACE.

started, it was found that a great loss of gold resulted, it being carried over with the zinc and zinc-oxide fumes in a practically non-condensable form. This was probably due to two causes : (1) Actual combustion of zinc into oxide, and mechanical removal of gold in the dense oxide fumes,—a defect subsequently remedied by inclusion of carbon in the charge, and by carrying out the distillation in a strictly reducing atmosphere ; and (2) loss due to the zinc vapour first formed at the bottom of the retort ascending through colder layers of zinc-gold powder in the upper portion of the charge, and giving rise to a kind of ebullition, whereby some of the rich product was blown over from the retort.

A simple method for overcoming the latter difficulty was

found by Picard in thoroughly moistening the precipitate before distillation with a strong solution of sugar, or similar cheap carbonaceous material, and making the mass into balls, which, after drying, form hard non-dusting cakes. The object of "balling" the charge is to afford free channels for

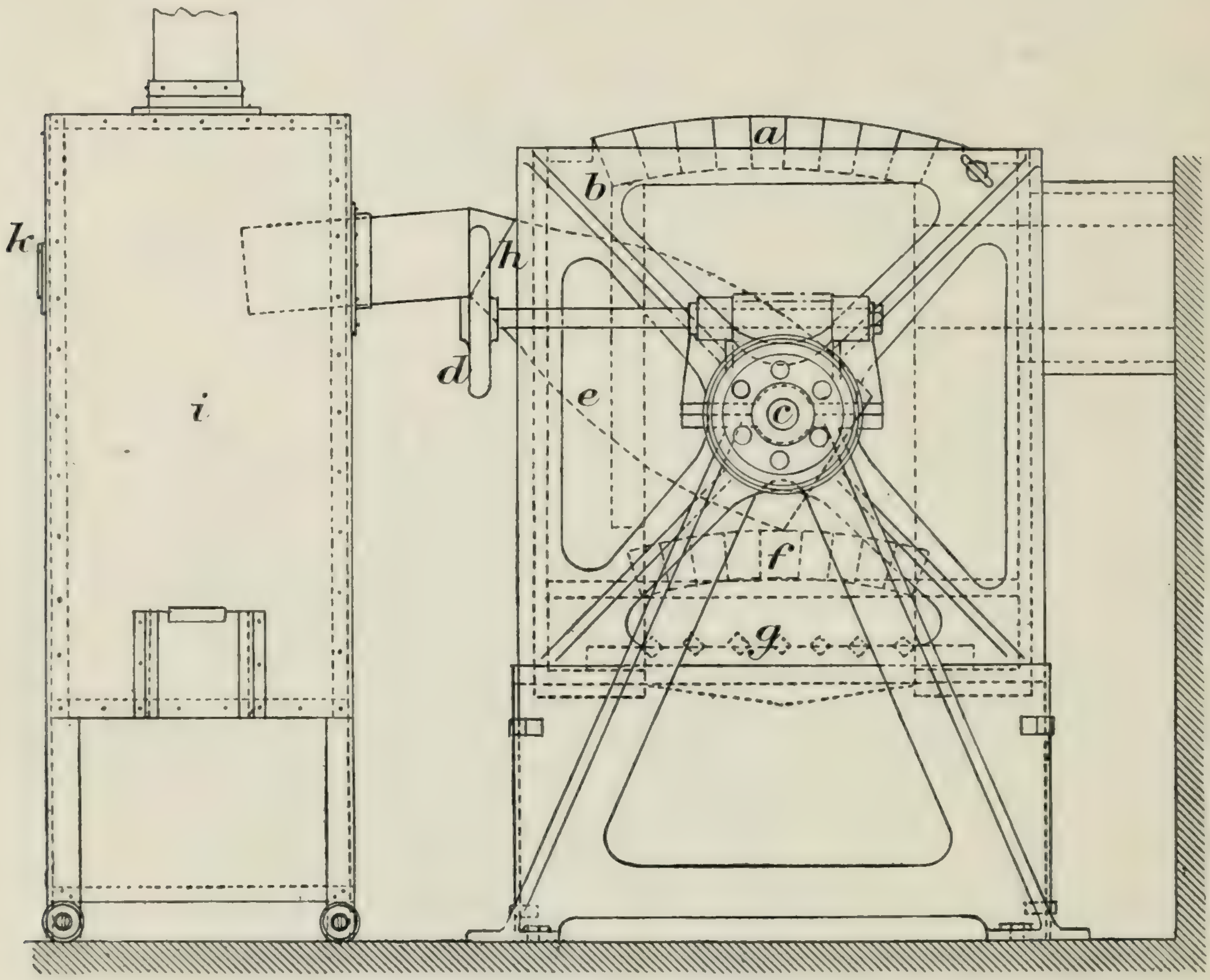



FIG. 179.—BALBACH FURNACE.

the exit of the metallic zinc vapours. Sugar or molasses is preferred, both on account of its cheapness, and because on heating in the retort it yields a hard, coherent, but open coke, which mechanically binds the gold; it further tends to promote the reducing atmosphere in the retort, preventing the formation of zinc oxide, and rendering additional fuel unnecessary. By checking production of all but traces of zinc oxide in the retort, difficulty in final smelting of the bullion is entirely avoided. Addition of a small quantity of borax to the sugar solution facilitates the running together of the gold globules during distillation.

The retort in use takes a charge of 65 to 70 lb. balled

precipitate. Its life is usually four operations, and it is turned one-quarter round on its bearing after each distillation, so as to ensure uniform wear. The time of distillation, from lighting

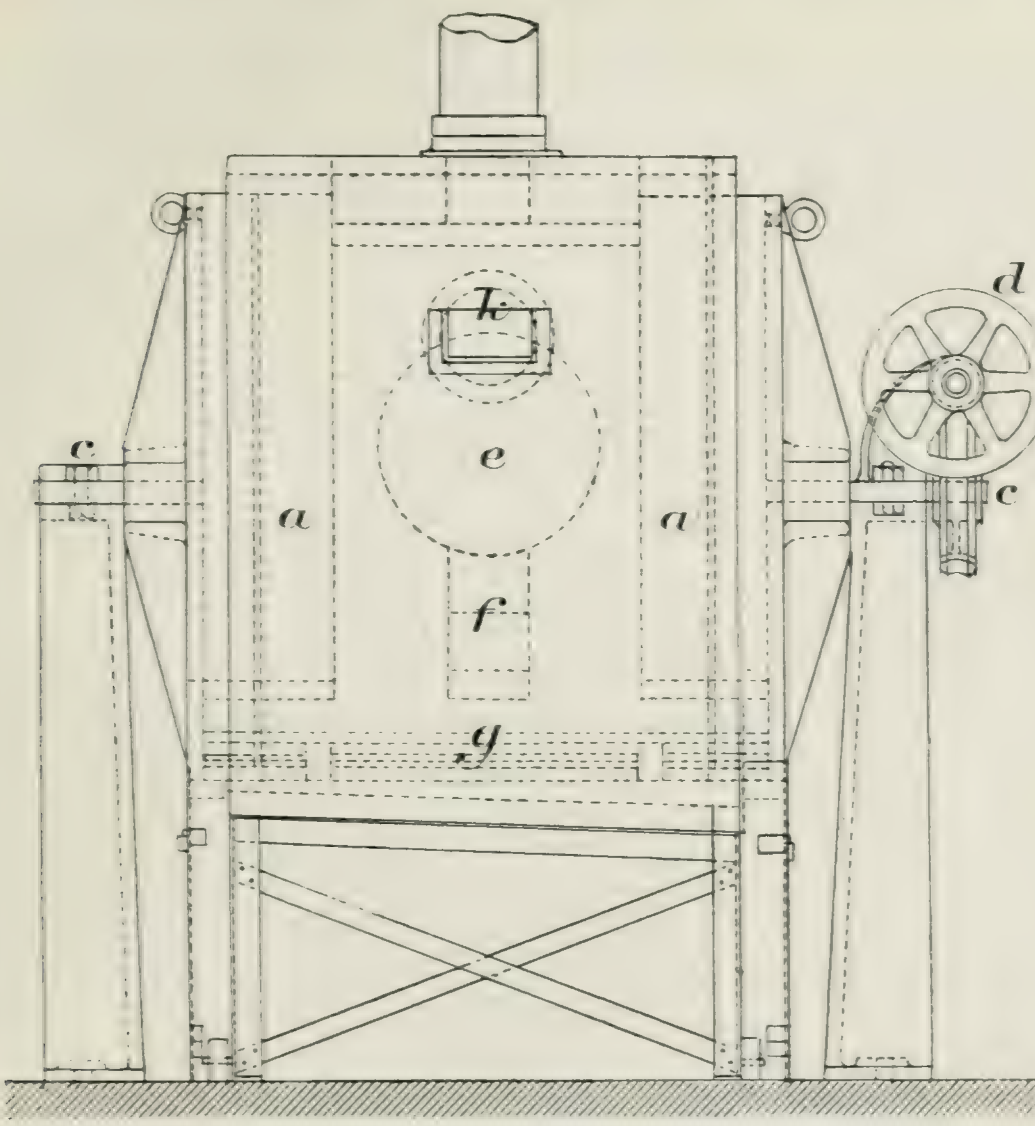



FIG. 180.—BALBACH FURNACE.

the fire to casting the bullion, is 12 hours, and the coal consumption is $2\frac{3}{4}$ cwt.

These figures, however, were reduced considerably by the employment of a retort with an annular shoulder moulded just inside the mouth, in order to prevent the drops of molten zinc, which condense somewhat readily in this region, from running back into the retort, and thus greatly prolonging the operation. The molten zinc is drained from the recessed depression through a small aperture in the lowest part of the retort-neck.

A plain sheet-iron condenser *i* is fixed to the mouth of the retort, to collect the zinc and zinc oxide which forms largely. A window *k* fixed in the front of the condenser, opposite the

mouth of the retort, permits of the operation being watched, and any deposit of zinc oxide which forms in the mouth of the retort can be removed by a rake. When distillation is complete, the condenser is detached, and the contents of the retort are cast into a mould by simply tipping the furnace.

An ingot of bullion, containing 80 to 90 % of the gold, is thus obtained, with a covering of thick pasty slag of 3 to 5 lb. weight. This slag consists of the residual carbon from the sugar, the borax, some graphite, a little zinc oxide, etc., which holds up the balance of the gold in "prills." It is not thought advisable to liquefy this slag in the retort itself, by the addition of suitable fluxes, as fluid slags would cause undue wear of the retort-walls. The slag is carefully scraped out, and is smelted for its gold-contents in an ordinary graphite crucible, a suitable mixture being: slag, 4 parts; soda-ash, 1 part; borax, $1\frac{1}{2}$ parts; and sand, $\frac{1}{2}$ part. The bullion obtained from this smelt is remelted with the metal cast direct from the retort, yielding the final gold-brick.

The fineness of the bullion obtained varies from 750 to 800, the impurities being chiefly zinc (10 %) and copper. The recovery-cost per oz. of gold, with a moderately rich precipitate, is said by Picard to rarely exceed 3*d.*

The distillation cost is as follows:—

| | *s.* | *d.* |
|---|---|---|
| Coal for distillation, $2\frac{3}{4}$ cwt. . . . . . | — | |
| ,, for smelt, $\frac{3}{4}$ cwt. . . . . . . . . | — | |
| ,, total $3\frac{1}{2}$ cwt., say . . . . . . . . | 3 | 6 |
| Quarter share of retort . . . . . . . . | 7 | $3\frac{1}{2}$ |
| Fluxes and crucible for final smelt . . . . | 4 | 0 |
| Total . . . . . . . . . . | 14 | $9\frac{1}{2}$ |

The gold obtained as actual bullion agrees with the extraction as shown by liquor and precipitate assays; the loss during the whole distillation is estimated at ·05 %.

The final slags are allowed to accumulate, and are periodically treated by the process suggested by Caldecott (see p. 694). This ingenious method answers perfectly with such products, and in both cost and efficiency is far superior to the usual method of crushing and panning.

The scale of operations at Lucknow was too small to afford

much basis for drawing parallels with other methods, but so far as it went, the idea of using a minimum of zinc dust as a precipitant, with thorough agitation, and intermittent discharges of about 4 t. every ½ hour into a centrifugal separator, seemed eminently satisfactory, treatment with sulphuric acid being made to follow for removal of the small excess of zinc.

*Lead.*—In the Siemens-Halske process, the frames carrying the lead cathode are taken out one at a time; the lead is removed, and replaced by a fresh sheet, and the frame is returned to the box, the operation taking a few minutes for each frame. The ordinary working is not interrupted, and cleaning out the boxes is only required at long intervals. The lead sheets are transferred to a cupelling furnace, to be separated into bullion and litharge. The operation requires special skill, and is always one involving considerable trouble, while the bulk to be dealt with is great, and the resulting litharge is very impure and of no great market value. The mixture going to cupellation generally contains only 2 to 5 % of gold.

*Charcoal.*—The gold-impregnated contents of the charcoal boxes is conveyed to furnaces, where the carbon may be slowly burned away. Sometimes these are small reverberatories, with about 6 side-charging doors to a length of 10 ft., each perforated with a number of holes to admit air under control for aiding the burning off. When the furnace is charged, a wood fire is lighted in the fire-box until the charcoal is well alight, when the dampers are closed, and it is allowed to slowly burn itself out. An improvement on the reverberatory is the Turnbull furnace, a small portable arrangement with condensing apparatus for arrest of values mechanically carried over by the draft, which has come into much favour in Victoria. In really large plants, however, the immense volume of charcoal to be dealt with would necessitate most extensive furnace accommodation. In any event, the burning away is not complete, and the furnace product is screened in a 30-mesh trommel enclosed in an iron box, recovered charcoal being returned to the furnace with the next charge, while the ash, containing about 5½ % bullion, goes to the melting-pot. As a rule, no

trouble is taken to reduce the bulk of melting charge and save fluxes and crucibles by leaching the ash with warm water, despite the fact that much of the bulk consists of soluble alkaline salts.

**Arrangement of Works.**—Cyanide works may be broadly divided into two distinct classes, those dealing with ores in bulk, and those dealing with tailings from other works; and the latter may be devoted either to tailings of current production, or to old accumulations, or to both. But in all cases there are certain conditions which must be fulfilled. The primary consideration is economy of labour and power in the conveyance of ore or tailings into and residues out of the works. Therefore where possible a site is chosen which possesses a suitable fall or grade to enable gravitation to be used to the utmost. But in many cases where tailings are treated these conditions are necessarily reversed, because the tailings heaps have already accumulated by gravitation upon the lowest point below the battery site, and no provision can be made at a lower elevation admitting of further handling and automatic dumping. Abundant accommodation for residues is, of course, a *sine quâ non*, with, in many cases, due precaution that they do not break away during heavy rain or flood, and cause damage to property or silting of water-courses. A sufficient water supply must be assured, with facilities for purification if necessary. Liquors of all kinds are, however, comparatively easy of control, being so readily and cheaply raised to any required elevation. Secure foundations are a matter of prime moment for all leaching and solution tanks, as they carry great weights, and the maintenance of a true level floor must be secured. At the same time a desideratum, and one worth some trouble to accomplish, is the arrangement of the various leaching (and settling) tanks in tiers. The total grade demanded will consist of—

(*a*) Fall of tailings launder computed at ½ to 1 in. per foot, according to coarseness of material and proportion of water to solids.

(*b*) Depth of hydrometric sizers for separation of slimes.

(*c*) Depth of settling tanks.

(*d*) Depth of supports of settling tanks.

(*e*) Depth of leaching tanks.

(*f*) Depth of supports of leaching tanks.

(*g*) Depth of filter for mechanical clarification of liquors.

(*h*) Fall for discharge of tanks, whether by manual labour and trucks from side or bottom, or by hydraulic sluicing through a launder.

Each works will require some combination of these steps, with figures governed by local considerations.

As to the housing of the plant, climatic conditions exert a controlling influence. In S. Africa and W. Australia, buildings are usually erected to cover the extractor boxes, furnaces, assay room, pumps and vacuum machinery, engines, and stores, while the leaching, settling, and liquor tanks are placed in the open, and suffer no harm beyond what may arise from excessive evaporation and the contamination due to dust storms. Where the weather is less consistent and more liable to spells of wet, the whole plant may be advantageously placed under a roof of timber and galvanised iron, with the sides left uncovered to such extent as is consistent with due regard to rain and prevailing winds, so as to secure abundance of light and ventilation. But in cool climates, such as New Zealand, full protection must be given to every portion of the works, both to obviate any possible risk of liquors freezing in pumps, pipes, or tanks, and because cyanidation is much impeded by low temperatures (see p. 482); and in countries liable to extremely severe weather, such as Utah, Dakota, and Colorado, most efficient means of maintaining heat throughout the plant is absolutely essential to proper results.

**Working Results.**—While the proposition at each and every mill is sure to possess some distinctive feature which militates against exact copying of another plant, yet examples of the methods employed and working results achieved on a number of important mines in various parts of the world cannot fail to be interesting and instructive. The appended tables, pp. 660–3, speak for themselves, and, though manifestly incomplete, convey much information in a small space.

CYANIDING: DETAILS OF COST PER TON TREATED.

| Mill. | Year. | Wages. | Fuel. | KCy. | Zn. | Lime. | Pb. | Royalty. | Sundries. | Total. | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| DRY-MILLED ORE: | | *d.* | *d.* | *d.* | *d.* | *d.* | *d.* | *d.* | *d.* | *s.* | *d.* |
| Luipaard's Vlei, S.A. | .. | 18·38 | 3·30 | 7·70 | 1·08 | 1·15 | .. | .. | 7·90 | 3 | 3·51 |
| Waihi, N.Z. | 1899 | 9·30 | | 27·60 | | .. | .. | 5·18 | 6·96 | 4 | 1·04 |
| TAILINGS: | | | | | | | | | | | |
| French Rand, S.A. | 1898 | 16·25 | 2·50 | 14·00 | 1·50 | ·50 | .. | .. | 3·00 | 3 | 1·75 |
| Witwatersrand, S.A. | 1898 | 10·75 | 2·00 | 8·50 | .. | .. | .. | .. | 14·50 | 2 | 11·75 |
| Standard, U.S. | 1895 | 27·11 | 5·23 | 12·60 | 1·28 | 5·50 | .. | .. | 1·61 | 4 | 5·33 |
| Harquahala, U.S. | 1897 | 22·47 | 8·44 | 15·00 | ·27 | ·29 | .. | 5·00 | 18·18 | 5 | 9·65 |
| SANDS: | | | | | | | | | | | |
| Myalls, N.S.W. | 1897 | 15·75 | 1·50 | 12·25 | 1·50 | ·50 | .. | .. | ·75 | 2 | 8·25 |
| Crown Reef, S.A. | 1898 | 10·83 | 2·15 | 19·72 | | | | 1·67 | 1·36 | 2 | 11·73 |
| Geldenhuis, S.A. | 1898 | 5·00 | 1·00 | 12·00 | 1·00 | ·50 | .. | .. | 12·00 | 2 | 7·50 |
| Jumpers, S.A. | 1899 | 8·50 | 1·50 | 13·50 | 2·00 | ·50 | .. | .. | 15·00 | 3 | 5·00 |
| Pioneer, S.A. | 1898 | 18·75 | ·25 | 12·75 | | | | .. | 9·75 | 3 | 5·50 |
| SLIMES: | | | | | | | | | | | |
| Crown Reef, S.A. | 1898 | 17·12 | .. | 23·03 | | | | ·43 | 2·56 | 3 | 7·14 |
| Geldenhuis, S.A. | 1898 | 9·25 | 2·00 | 3·50 | .. | 5·50 | 5·00 | .. | 6·00 | 2 | 7·25 |

CYANIDING ORE: GENERAL RESULTS.

| Mill. | Year. | Assay. | Residues. Assay. | Recovery. | | Cost per ton. | |
|---|---|---|---|---|---|---|---|
| New Zealand: | | dwt. | dwt. | dwt. | % | *s.* | *d.* |
| Talisman . . . | 1899 | 16·07 | 3·06 | 13·01 | 81 | 11 | 9·5* |
| Waihi . . . . | 1899 | .. | .. | .. | 88 | 3 | 7·86 |
| Transvaal: | | | | | | | |
| Afrikander . . | .. | .. | .. | .. | 79† | .. | |
| Lisbon Berlyn . | .. | 10·00 | .. | .. | 70 | .. | |
| Luipaard's Vlei . | .. | 8·28 | 1·4 | 6·80 | 82 | 3 | 5·86 |
| United States: | | | | | | | |
| Mercur, Utah. . | .. | 15·00 | .. | .. | 73 | 10 | 0·00* |
| Republic, Wash. . | 1897 | 66·00 | 6·6 | 59·4 | 90 | 20 | 10·00 |

CYANIDING TAILINGS: GENERAL RESULTS.

| Mill. | Year. | Assay. | Residues. Assay. | Recovery. | | Cost per ton. | |
|---|---|---|---|---|---|---|---|
| Australia: | | dwt. | dwt. | dwt. | % | *s.* | *d.* |
| Bayley's United . | 1898 | 13·14 | ·79 | 12·35 | 94 | 8 | 3·00 |
| Myall's . . . . | 1898 | 2·74 | ·37 | 2·34 | 85 | 2 | 8·62 |
| India: | | | | | | | |
| Champion Reef . . | 1898 | .. | .. | 3·71 | 56 | 3 | 0·75 |
| Mysore . . . . | 1897 | 4·02 | 1·41 | 2·61 | 65 | 2 | 6·80 |
| Do. . . . . | 1898 | 3·42 | 1·27 | 2·15 | 63 | 2 | 8·25 |
| Nundydroog . . | .. | .. | .. | 2·01 | .. | 2 | 4·50 |
| Transvaal: | | | | | | | |
| Driefontein . . . | 1899 | 6·20 | 1·46 | 5·17 | 83 | 2 | 11·00 |
| French Rand . . | 1898 | .. | .. | 4·80 | .. | 3 | 1·75 |
| New Comet . . . | 1899 | 5·60 | 1·18 | 3·89 | 69 | 3 | 7·00 |
| New Goch . . . | .. | .. | .. | 4·16 | .. | 3 | 11·25 |
| Paarl . . . . . | .. | .. | .. | 4·06 | .. | 2 | 6·50 |
| Robinson . . . | 1898 | 7·06 | 1·61 | 5·45 | 77 | 2 | 8·03 |
| Windsor . . . . | 1898 | 5·00 | 1·06 | 3·94 | 79 | 3 | 11·50 |
| Witwatersrand . . | .. | .. | .. | 3·18 | .. | 2 | 11·75 |
| United States: | | | | | | | |
| Harquahala . . . | 1897 | 4·47 | 1·25 | 3·22 | 72 | 5 | 9·65 |
| Montana . . . . | 1898 | .. | .. | .. | .. | 7 | 0·00‡ |
| De Lamar, Idaho . | 1899 | 6·36 | 1·65 | 4·71 | 74 | 6 | 6·61 |
| Standard . . . | 1895 | 5·20 | 1·69 | 3·40 | 65 | 5 | 8·65 |

* Includes milling. † 86% when plant was new, but fell to 72%.
‡ Includes hauling.

## CYANIDING SANDS: GENERAL RESULTS.

| Mill. | Year. | % of tons milled. | Assay. | Residues. Assay. | Recovery. | | Cost per ton milled. | |
|---|---|---|---|---|---|---|---|---|
| Transvaal:— | | | dwt. | dwt. | dwt. | % | s. | d. |
| Angelo . . . . | 1899 | 79 | 8·37 | 1·61 | 7·10 | 85 | 3 | 0 |
| Bonanza . . . . | 1899 | 62½ | 10·88 | 1·56 | 9·32 | 86 | 5 | 9† |
| Crown Deep . . . | 1898 | 82½ | 5·85 | .. | 4·58 | 78 | 2 | 10† |
| Durban Roodeport . | 1898 | 59½ | 5·39 | 1·33 | 4·06 | 75 | 3 | 2½ |
| Do. Deep . . . | .. | .. | .. | .. | 4·84 | .. | 2 | 1½ |
| Ferreira . . . . | 1898 | 71¼ | 7·89 | 1·48 | 6·41 | 81 | 3 | 6 |
| Geldenhuis . . . | 1898 | .. | 5·31 | .. | 4·13 | .. | 2 | 7½ |
| Do. Deep. . | 1898 | 74½ | 5·18 | .. | 3·59 | 69 | 2 | 3½† |
| Glen Deep . . . | 1898 | 63¼ | 5·60 | .. | 4·68 | 84 | 2 | 11½† |
| Glynn's Lydenburg . | 1899 | 63 | .. | .. | 8·44 | .. | 4 | 1* |
| Jumpers . . . . | 1899 | 63½ | 3·88 | ·74 | 3·15 | 81 | 3 | 5 |
| Do. Deep . . | 1898 | 69¼ | 6·33 | .. | 4·91 | 77 | 3 | 2½† |
| Langlaagte Deep. . | 1898 | 69½ | 4·63 | .. | 3·37 | 73 | 2 | 10½† |
| Nourse Deep . . . | 1899 | 71½ | 5·60 | .. | 4·24 | 76 | 3 | 7† |
| Pioneer . . . . . | .. | .. | .. | .. | 4·76 | .. | 3 | 5½† |
| Robinson . . . . | .. | .. | .. | .. | 8·67 | .. | 4 | 2½† |
| Rose Deep . . . | 1898 | 74½ | 5·74 | .. | 4·96 | 77 | 2 | 5 |
| Sheba . . . . . | .. | .. | .. | .. | 4·31 | .. | 1 | 6½ |
| Transvaal Estates . | 1898 | .. | .. | .. | 6·04 | .. | 5 | 2½* |
| Van Ryn . . . . | .. | .. | .. | .. | .. | .. | 2 | 4¾ |
| Village . . . . | .. | .. | .. | .. | 4·11 | .. | 2 | 1½ |

## CYANIDING CONCENTRATES: GENERAL RESULTS.

| Mill. | Year. | Concentrators. | % of tons milled. | Assay. | Residues. Assay. | Recovery. | | Cost per ton milled. | |
|---|---|---|---|---|---|---|---|---|---|
| Italy: | | | | dwt. | dwt. | dwt. | % | s. | d. |
| Pestarena | 1898 | .. | .. | 9·73 | .. | 7·94 | 82 | .. | |
| Transvaal: | | | | | | | | | |
| Bonanza . | 1899 | .. | 12 | 20·06 | 3·07 | 16·99 | 85 | 5 | 9‡ |
| Geldenhuis | 1898 | Troughs | .. | 13·97 | .. | 10·84 | 77 | 3 | 4 |
| Jumpers . | 1899 | Do. | 5¼ | 15·38 | 2·39 | 13·86 | 90 | 3 | 5‡ |
| Sheba . | .. | .. | .. | .. | .. | 128·67 | .. | 67 | 7·5 |

* Per ton treated. † Mixed with concentrates.
‡ Mixed with sands.

CYANIDING SLIMES: GENERAL RESULTS.

| Mill. | Year. | % of tons milled. | Assay. | Residues. Assay. | Recovery. | | Cost per ton milled. | |
|---|---|---|---|---|---|---|---|---|
| India: | | | dwt. | dwt. | dwt. | % | *s.* | *d.* |
| Mysore | 1898 | .. | 2·05 | ·75 | 1·30 | 63 | | .. |
| Ooregum | 1898 | .. | 3·12 | 1·00 | 2·12 | 68 | | .. |
| Transvaal: | | | | | | | | |
| Bonanza | 1899 | 25 | 6·21 | ·89 | 5·32 | 86 | 9 | 5 |
| Crown Deep | 1898 | 16 | 5·85 | .. | 3·43 | 59 | 3 | 0¾ |
| Durban Roodeport | 1898 | 27 | 5·39 | 2·19 | 3·20 | 59 | | .. |
| Ferreira | 1898 | 25 | 4·89 | .90 | 2·70 | 55 | 6 | 6* |
| Geldenhuis | 1898 | .. | 3·81 | .. | 3·51 | .. | 2 | 7¼ |
| Do. Deep | 1898 | 25 | 2·93 | . | 1·03 | 35 | 2 | 11½ |
| Glen Deep | 1898 | 30 | 2·81 | .. | 2·04 | 73 | 2 | 10½ |
| Glynn's Lydenburg | 1899 | 31 | .. | .. | 8·14 | .. | 6 | 5½* |
| Jumpers Deep | 1898 | .. | 3·50 | .. | 1·52 | 44 | 2 | 9 |
| Langlaagte Deep | 1898 | 18½ | 1·97 | .. | 1·35 | 60 | 5 | 8 |
| Nourse Deep | 1898 | .. | 4·77 | .. | 2·72 | 57 | 3 | 2½ |
| Do. | 1899 | 28½ | 2·80 | .. | 1·89 | 68 | | .. |
| Rose Deep | 1898 | .. | .. | .. | 2·31 | 63 | | .. |
| Sheba | .. | .. | .. | .. | 1·89 | .. | 7 | 1 |
| Transvaal Estates | 1898 | .. | .. | .. | 3·40 | .. | 5 | 5¼* |
| Van Ryn | 1899 | .. | 3·50 | 1·10 | 2·40 | 68 | 3 | 0* |
| Village | .. | .. | .. | .. | 2·49 | .. | 4 | 0½ |

*Australia.*—In all the Australian Colonies cyanidation has become of wide application for treating tailings, and in rare instances it deals with ore direct.

A slimes plant for agitation only, in course of erection by the author at Lucknow, New South Wales is illustrated in Fig. 181. The inclined tramway conveys the slimes (an old accumulation) to a platform, whence they are fed into a trommel revolving in water, and are thus reduced to pulp which can be centrifugally pumped into the row of steel vats; these latter are provided with agitating mechanism which can be raised and lowered at will. Discharge will be by

* Per ton treated.

means of water and agitation through side doorways into a launder.

At the Myall's United mill, N.S.W., according to W. R.

FIG. 181.—SLIMES PLANT AT LUCKNOW, N.S.W.

Thomas,* the tailings from a wet-crushing battery, after leaving plates and blanket strakes, are raised into 4 settling tanks, where the slimes, amounting to "only a small percentage" (quantity and value not given), which hitherto have not been treated, are removed. The sands pass to the cyanide plant, consisting of 8 leaching vats (44·8 t. capacity), 3 solution tanks, and a store sump. The vats are of wood, with round-iron bands fitted with tightening screws. There are 4 zinc boxes and a liquor pump. The plant can treat 2464 t. per month (28 days), and cost 2088*l.* The period of contact with strong solution is 12 to 14 hours. Residual tailings assay ·372 dwt. per ton. Cyanide costs 1*s.* 2*d.* per lb. The monthly figures of cost of cyaniding range from 2*s.* 7·88*d.* to 3*s.* 0·96*d.* per ton; the average for 6 months is put at 2*s.* 10*d.*, but the analysis of cost works out at 2*s.* 8·62*d.*, thus:—

| | *s.* | *d.* | | *s.* | *d.* |
|---|---|---|---|---|---|
| Labour (filling vats) | 0 | 10·75 | Coke | 0 | ·54 |
| Supervision | 0 | 3·30 | Chemicals | 0 | ·45 |
| Boys | 0 | ·89 | Zinc | 0 | 1·51 |
| Labourers (extra) | 0 | ·89 | Cyanide | 1 | 0·23 |
| Tip-timber | 0 | ·09 | Sundries | 0 | ·45 |
| Steam | 0 | 1·07 | | | |
| Lime | 0 | ·45 | Total per ton | 2 | 8·62 |

The Kalgurli field (W. Australia) is remarkable for the development of filter-press treatment, forced upon mill managers by the limited and impure water supply. Some plants are crushing wet and amalgamating, applying cyanide to the tailings; while others are adopting cyanidation for the ore in mass, either dry-crushed and roasted, or wet-crushed and employing the Sulman process. Cyaniding cost on wet-crushed sands at four mills is said to average 4*s.* 8*d.* per ton; and filter-press treatment of slimes, 7 to 9*s.* per ton.

*India.*—The Colar mills have a very simple product to deal with in the battery tailings, which until quite recently underwent the old-fashioned process of grinding and amalgamation in Wheeler pans. The Champion Reef Co. uses

* Trans. Inst. M. & M., vii. 145 (1899).

20-mesh screens in the battery, and from these coarse tailings only 56 % is recovered, with a consumption of 1 lb. KCy per ton. The Mysore* plant, with 40-mesh screens, and an average tailings assay of 3·42 dwt., gets an extraction of 2·16 dwt. (residues assay 1·26 dwt.), or 63 %, the cyanide consumption being 1·54 lb. per ton, and zinc waste ·205 lb. per ton. The slimes works dealt with over 30,000 t. carrying 2·04 dwt. per ton, and gave an extraction of 1·3 dwt. (residues assay ·74 dwt.) or 63 %, the consumption of KCy being 1·05 lb. and of zinc ·116 lb. per ton. The cost in 1897 was quoted by L. Pitblado † at 2*s.* 6¾*d.* per ton.

*New Zealand.*—The earliest New Zealand mills relied on copper-plate amalgamation, and lost more than half the bullion in the ore. They were succeeded by dry crushing and pan amalgamation, which reached 60 % recovery. Next cyanidation of dry-milled ore followed, extracting about 80 %. Finally, wet milling is becoming established again, with amalgamation for recovery of the coarse free gold, and cyanidation coupled in some instances with concentration for the tailings, the best returns exceeding 90 %.

At the Crown mill, the original practice was kiln-drying and dry-stamping, the cyanide plant consisting of 24 wooden (kauri) vats, 11 × 9 × 3 ft. 9 in. deep; also 14 agitators, 8 of which are 5 ft. × 4 ft. 9 in. diam., and 6 are 6 ft. × 5 ft. 6 in. diam. The agitators, however, were seldom used, preference being given to percolation. There are 3 concrete sumps, 15 × 12 × 6 ft. deep. The dry-milled ore was discharged upon an 8-in. rubber belt with rope edges, and conveyed to and across a hopper 110 ft. long, running the entire length of the cyanide plant house. This hopper had 20 doors for discharging the sand into trucks, which were then run straight out over the percolating vats into travellers running on rails, which are fitted with hand-traversing gear, enabling a truck to be tipped at any part of the vat.

Now circular steel vats, 22 ft. diam. × 4 ft. deep, are employed, the filter bottoms occupying 5 in. of the depth.

* Official Report for 1898.

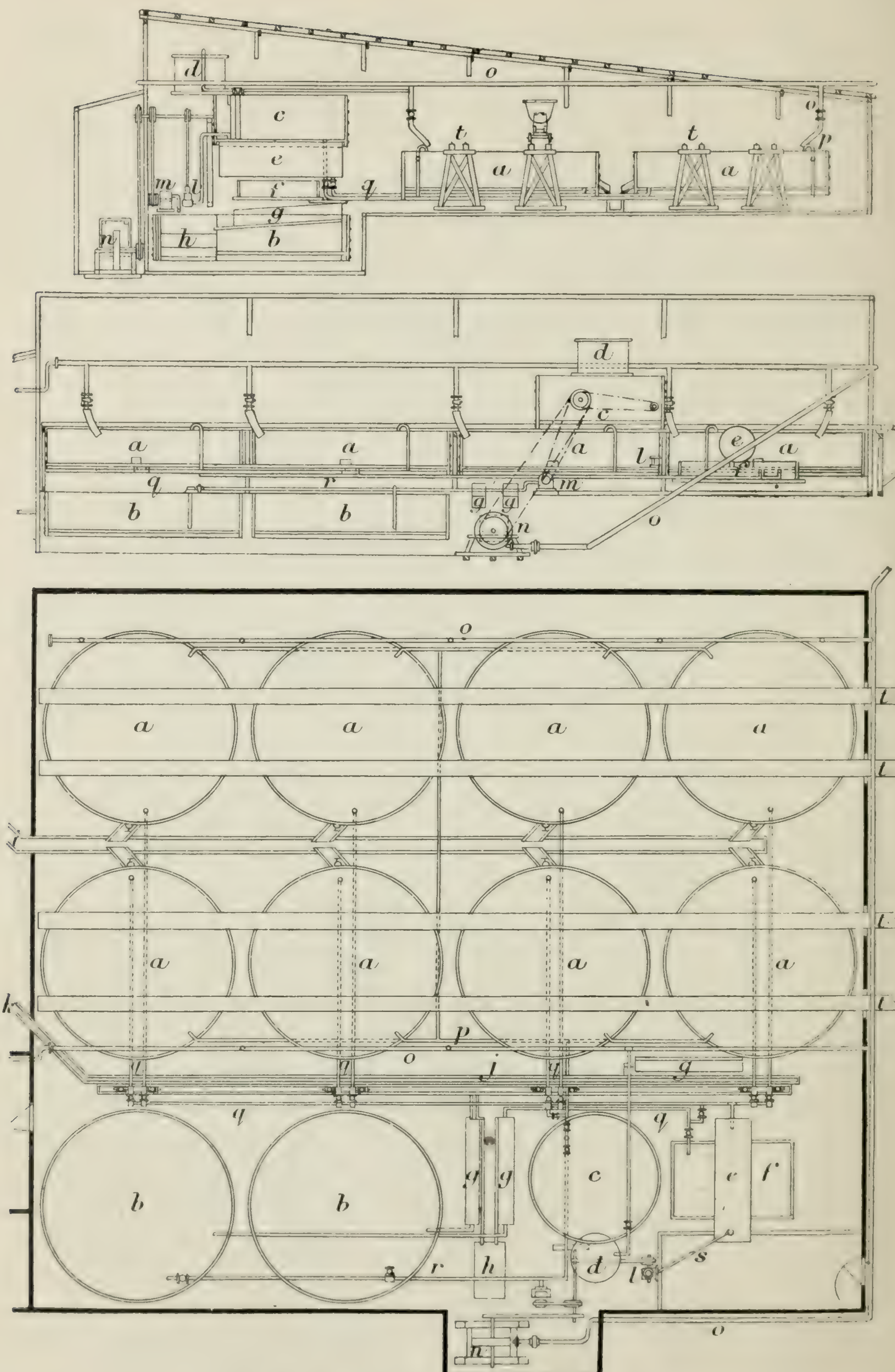
† Jl. Soc. Chem. Ind., Feb. 28, 1898.

In 1897, a commencement was made with wet crushing, using weak cyanide liquor instead of water, at the rate of about ½ t. solution to the ton of ore, and adding strong liquor in the leaching tanks. The whole stream of pulp (sands and slimes together) is conducted by a launder to one vat, and allowed to discharge into the centre until the vat is half full. It is then conveyed to another vat, which is filled in the same manner. The mixed sands and slimes in the first vat are allowed to settle for an hour or two, after which the fairly clear top solution is siphoned off into a collecting tank, whence it is pumped up to two elevated tanks, from which the solution for the battery is supplied. The pulp is again diverted into the first vat until its charge is complete. After settling, the clear top solution is again drawn off. In this way several vats are in course of filling at the same time. Deposition of slimes is effected without the aid of lime, by allowing the solution to percolate from the bottom of the vat during the filling and period of settlement. The downward currents promoted by draining from below are said to be very effective in this respect, but the charge is not over 30 in. deep.

The Komata Reefs has also recently been converted from dry to wet milling. The vats have been fitted with an internal rim launder for overflow, and the water reaches the same by filtering through a screen of filter-cloth. This has proved successful in retaining all the sands and slimes. By using a revolving distributor and filling into water-full vats, satisfactory percolation is attained, and extraction is claimed to be 93 %, or slightly higher than previously.

Statistics of the Talisman for 1899 show that the assay value of the ore is 16·07 dwt. gold and 6 oz. 16·01 dwt. silver; recovery 13·01 dwt. (81 %) and loss 3·06 dwt. of the gold, and 3 oz. 13·03 dwt. (53·7 %) and 3 oz. 2·98 dwt. respectively of the silver. The costs for milling and cyaniding, on 11,014 t., were 11*s.* 9½*d.* per ton. Cyanide consumption was 3·09 lb. per ton, costing 1*s.* 3*d.* per lb.

Figs. 182 to 184 represent the Cassel works of the Waihi Co., erected for treating tailings: *a*, leaching tanks; *b*, sumps; *c*, storage tank; *d*, KCy dissolving tank; *e*, vacuum chamber;



FIGS. 182 TO 184.—CASSEL WORKS, WAIHI.

*f*, tank to receive contents of *e*; *g*, zinc boxes; *h*, slime settler; *i*, discharge launder; *j*, strong-liquor launder to zinc box; *k*, waste launder; *l*, air pump; *m*, centrifugal pump; *n*, Pelton wheel motor; *o*, fresh-water pipes; *p*, KCy pipes from storage tank to leaching tanks; *q*, weak-liquor pipes from leaching tanks to vacuum chamber and from vacuum chamber to zinc boxes; *r*, pump pipes; *s*, air-pump pipes; *t*, tramways over tanks. This installation deals with tailings from pan amalgamation consisting largely of slimes and carrying their values in the shape of amalgam. The course of treatment adopted, according to E. G. Banks, chemist in charge, is as follows:—

| | Hours. |
|---|---|
| Filling vat, 30 t., 3 men | 8 |
| Preliminary lime or water-wash, 6 t., with vacuum | 6 |
| Leaching, strong solution, 8 t. ·6 % KCy | 30 |
| ,, weak solution, 4 t. ·2 % KCy (from strong sump) | 12 |
| Washing (using vacuum) with wash from weak sump, 3 washes of 4 t. each | 36 |
| Water-wash, 4 t. | 12 |
| Discharging vat, one man sluicing | 4 |
| Total | 108 |

The plant at the old Waihi mill proper closely resembles that of the Crown Co., but their new mill at Waikino possesses some novel features. The vat house contains 10 concrete (well tied in with wire) rectangular vats, each 50 × 40 ft., and holding (when filled to a depth of 2 ft.) 150 t., or one day's supply. The pulp is delivered from the main conveyor to a rubber-belt elevator discharging on to a cross-conveyor in the roof, at right angles to the main axis of the building. This again supplies two conveyors, one in each roof-truss, running the length of the house and central with each row of 5 vats. These conveyors are of the same type as the main one connecting battery shed and vat house, and are driven by the same motor. Over each line of vats is a traveller fitted with traversing gear and hopper, so that the pulp may be spouted down from the overhead conveyor, evenly and gently distributed in the vats, and not pack. The filter-cloth is laid over a wooden-slat grating, and caulked down with rope. No

storage hoppers exist, as the capacious vats fulfil that object. Down the centre of the house is an alley-way for pipes, floored with cement, drains being provided to catch any leakage. The pipes are laid with running joints at intervals, for disconnection and removal of internal deposits. Zinc precipitation is used. The tailings are sluiced out with water. All mixing and running-on of liquors is done in daytime, to reduce night work to a minimum. The first solution is applied from below, in the proportion of $\frac{1}{3}$ t. (containing ·33 to ·4 % KCy) to the ton of ore. As soon as it begins to show on the surface, the remainder is run in at top, and what is left below the filter-bottom is pumped up. Percolation, assisted by a vacuum equal to 20 or 25 in. mercury, occupies 30 to 40 hours, when weak solutions (·1 to ·03 %) are applied, followed by water-washes (about 1 t. per ton). The time required for treating each vat totals $6\frac{1}{2}$ days. Cyanide consumption is $1\frac{2}{3}$ to 2 lb. per ton. Cyaniding costs at the new mill are as follows:—Labour, 4·25*d.*; KCy, 1*s.* 10·92*d.*; zinc, 1·92*d.*; filter-cloths, ·68*d.*; cleaning-up and smelting gold-mud, 3·91*d.*; total, 2*s.* 9·68*d.* per ton.

The Waitekauri mill, with dry crushing and cyanidation, gets an extraction of about 40*s.* from 50-*s.* ore, or 80 %. Owing to copper in the ore being precipitated with the gold, all the zinc in the extractor boxes is smelted, and the bullion is only about 230 fine.

At the new Woodstock mill, the chalcedonic quartz is wet-crushed to 40 mesh, passed over amalgamated copper plates, and elevated to a pointed box. The slimes from this pass to a slime tank, while the sands (containing some slimes and about 1 % of concentratable material) are run over vanners. The vanner-tailings are sent through a simple automatic distributor to the leaching vat. One vat is filled at a time, the overflow of slimes from this passing to the slimes tank. The slimes are treated by agitation, and when the gold is dissolved, lime is added, and the clear solution is decanted off. The sands and heavy slimes are treated by ordinary filtration. The concentrates, which average 30 to 40*l.* per ton, the larger value being silver sulphide, are treated with a 4 % solution of

cyanide for 36 hours, 2 lb. of lime being added to each ton of concentrates. The agitators hold about 1½ t., and the recovery is said to vary from 90 to 95 %, at a cost of 16*s.* per ton treated. The bullion is worth only 8 to 12*s.* per oz.

*Transvaal.*—In the ordinary practice of the Rand, the battery tailings carry 3 to 4 % of sulphides (chiefly iron pyrites) and assay ·3 to ·6 oz. gold per ton. The sulphides are generally eliminated by vanners or hydraulic boxes, and specially treated by chlorination or cyanidation, according to their grade ; the remainder of the pulp is then classified into sands and slimes for separate leaching. The concentrates are run into an elevated vat and allowed to settle, the water being then drained off. For a period of 5 days a ·3–·5 % cyanide solution is pumped through, after which it is drained off, and the tailings are discharged by a bottom door into another tank underneath, and are there treated in a similar manner for a further period of 10 to 20 days, with a ·15–·2 % solution of cyanide. This solution is then drained off, and the concentrates are washed with a very weak solution, allowed to drain, discharged by a bottom door into cars, and trammed to the waste-heap. The sands from the classifiers are allowed to settle in another set of tanks, and the water is drained off. They are washed with an alkaline solution, and then with a ·5 % KCy solution, which is pumped through for a period of 3 to 4 days. This solution being drained off, the material is treated for 2 days with a ·25 % solution ; the sands are then left in a state of quiescence for 3 hours, and the solution is allowed to drain off. The solids are next passed through the bottom-discharge door into the lower vat, where they are washed with the weakest solution available, drained, discharged into cars, and trammed to the waste-dump. The slimes are allowed to flow into a special set of tanks, one situated above the other, until one-fifth of the upper tank's capacity is filled with solid slimes. The water is then decanted off. By means of a centrifugal pump, a ·1 % KCy solution is played upon the settled slimes, and they are washed into the lower tank, which brings them thoroughly in contact with the solution. They are then allowed to settle, and the cyanide solution is

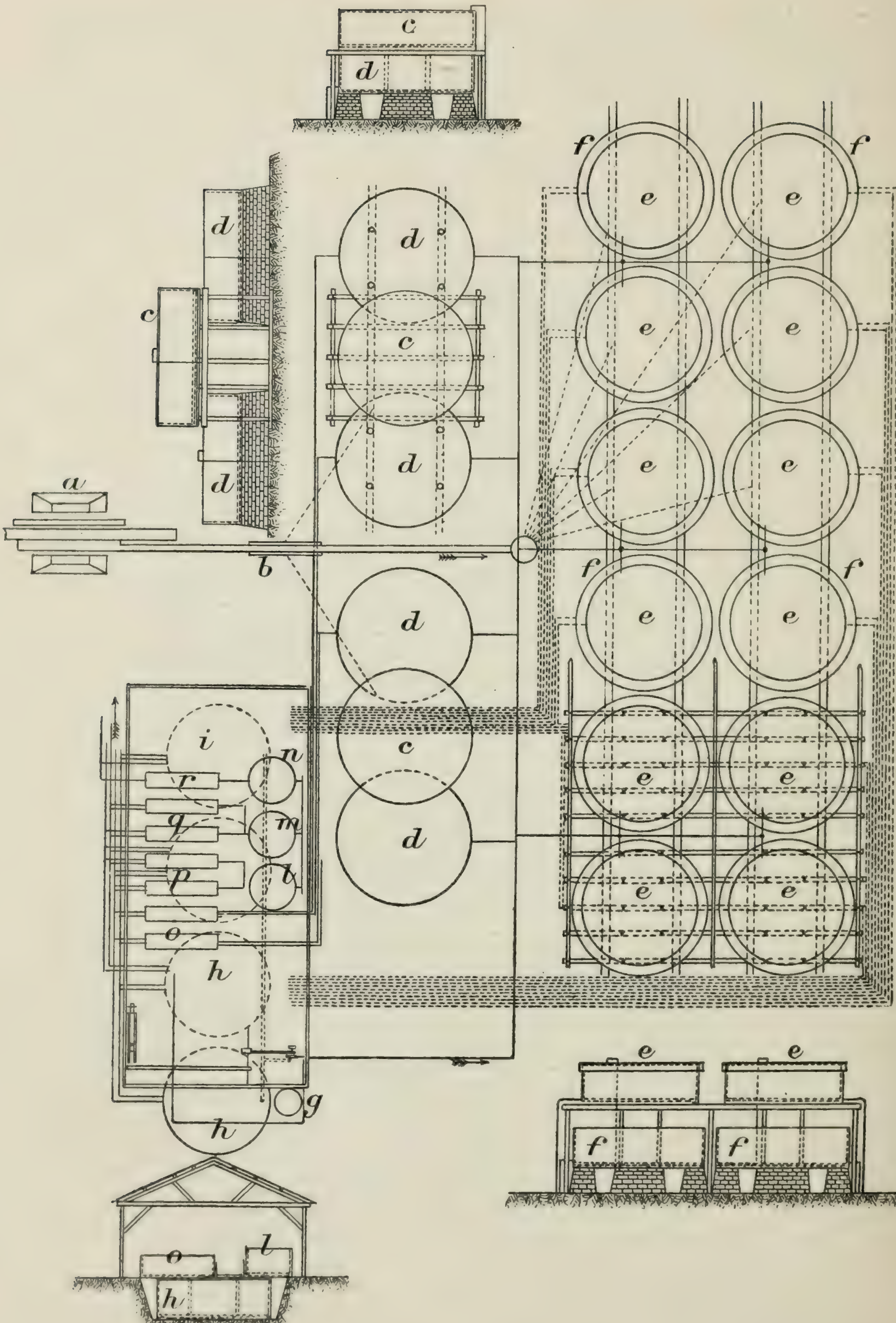



FIG. 185.—TYPICAL RAND PLANT.

decanted off. In the case of very rich slimes, this process is repeated, and they are transferred to a third tank, and thence trammed to the waste-dump. At some mines the tailings are run into settling ponds for storage, and when needed for treatment are carried to the vats by cars. The drawbacks to this are the cost of re-hauling the tailings, and the fact that the sulphurets will become partially oxidised, and will be left in a condition which makes them objectionable in cyanide treatment.

A typical Rand plant is shown in Fig. 185: *a*, tailings elevator wheels; *b*, classifiers for removing sulphurets; *c*, distributing concentrates; *d*, leaching concentrates; *e*, intermediate leaching sands; *f*, second leaching sands; *g*, KCy dissolving; *h*, strong liquor; *i*, weak liquor; *k*, waste liquor; *l*, *m*, *n*, strong, weak, and waste settling-tanks; *o*, *p*, *q*, *r*, precipitation boxes for concentrates, strong, weak and waste liquors respectively.

Fig. 186 shows a plan of a plant for treating slimes for the decantation method, as adopted in the Transvaal and designed by the Cyanide Plant Supply Co., London. The collecting vats are of steel, with conical bottom, and measure 38 ft. diam. by 10 ft. and 13 ft. deep; they have sub-level delivery and swivel-jointed solution decanting pipes, with pressure hose and nozzle for sluicing out. The mixing vat has apparatus for maintaining agitation and aëration.

Fig. 187 is from a photograph of the largest slimes plant in S. Africa, designed by the late Major Seymour.

The Angelo Co., in 1899, treated 98,783 t. of sands, and recovered in this way 36½ % of the value of the ore milled. The residues showed comparatively high value, owing to imperfection of classifying apparatus, and water highly charged with organic matter, which faults were being remedied. As yet, no slimes have been treated, but they are stored; they amounted to 21,783 t. for the year (17½ % of tonnage milled), and assayed 5·8 dwt. per ton.

At the Bonanza works some 3000 t. of high-grade tailings and concentrates are dealt with monthly. A plunger pump raises the battery pulp to hydraulic classifiers over the collecting

vats. The latter have automatic distributors and slimes gates ; the overflow passes to other classifiers, where some fine sand (leachable) is recovered and sent to the plunger pumps. There

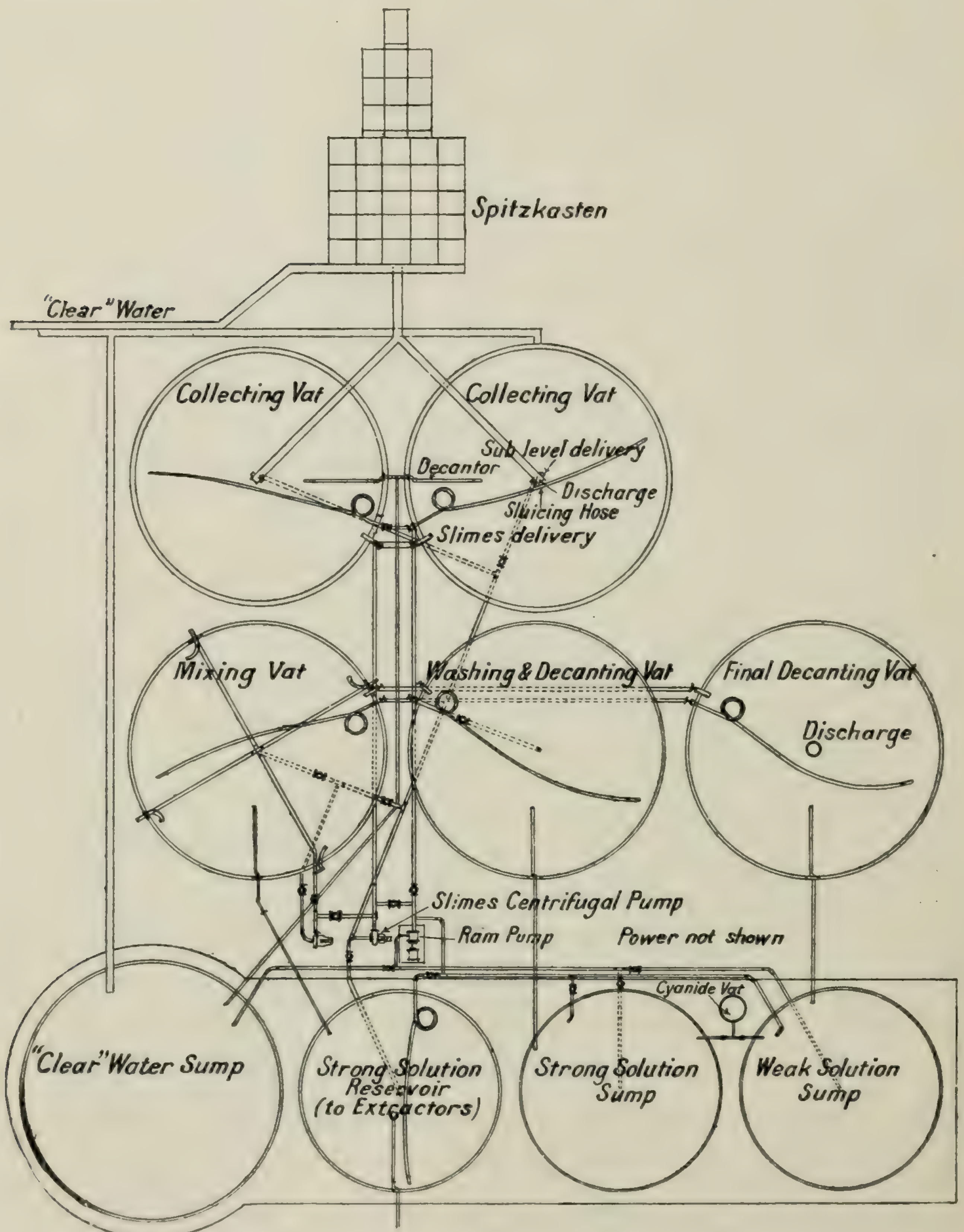



FIG. 186.—DECANTATION SLIMES PLANT.

are two tiers (6 in each) of 100-t. vats, placed one above the other. Two of the upper vats receive concentrates, the others taking general pulp. No more than 20 % of coarse material

is separated, or percolation and extraction of the remainder are much hindered. Strong liquors range from ·04 to ·1 %;

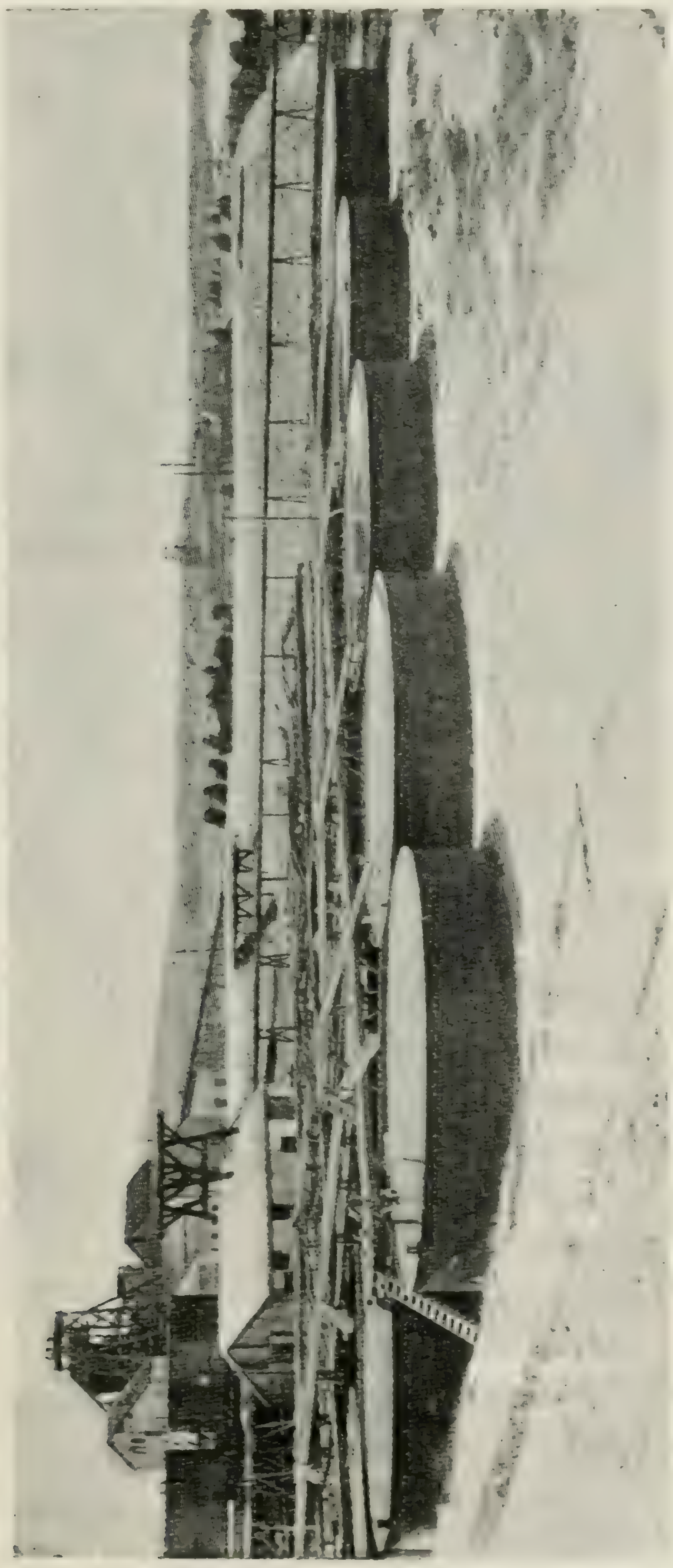


FIG. 187. DECANTATION SLIMES PLANT.

weak, from ·025 to ·03 %. Average extraction from sands is 81 %. Precipitation is by Siemens-Halske process: 2 strong and 2 weak boxes, each 30 × 4½ × 3 ft. deep, are connected

with a dynamo giving 200 amperes at 8 volts; each box contains 156 (1350 lb.) iron anodes covered with fine sacking, and 156 cathode frames holding 468 lb. lead foil cut into 1-in. strips and thus giving a total surface of 5990 sq. ft. Strong liquors flow at the rate of 3 t. per hour; weak, at 1 t.

In 1895, the City and Suburban plant had a capacity of 15,000 t. a month. The tailings were elevated by wheel to hydraulic classifiers, the sands assaying about 15 dwt. and the slimes 4 dwt. per ton. The treatment was by direct filling, and occupied 10 days, extraction reaching about 62 %. Discharged residues had a value of nearly 2 dwt. per ton. The consumption of chemicals per ton was ·649 lb. KCy, ·228 lb. zinc, and ·006 lb. caustic soda. Working costs were stated at 4*s.* 2¾*d.* per ton.

The Driefontein mill, in 1899, treated 138,117 t. of sands (80 % of tonnage milled) and recovered in this way 34½ % of the total value. The residues showed comparatively high value (1·46 dwt.) owing to difficulties of water supply both in quantity and quality. From sands assaying 6·2 dwt., an extraction of 5·175 dwt. fine gold (83½ %) was obtained.

In the Ferreira report for 1898, the working costs per ton milled are thus tabulated:—

| | *s.* | *d.* |
|---|---|---|
| Elevating tailings and settling sand | 0 | 2·004 |
| Transport to cyanide works | 0 | 2·841 |
| Charging treatment vats | 0 | 1·459 |
| Discharging ,, | 0 | 2·616 |
| Transport to waste-heap | 0 | 2·892 |
| General treatment of sands | 1 | 6·094 |
| | 2 | 5·907 |
| Pumping slimes to works | 0 | 3·393 |
| General slime treatment | 1 | 1·196 |
| Work on slime dams | 0 | ·380 |
| Royalty | 0 | 1·197 |
| | 1 | 6·167 |
| Grand total | 4 | 0·074 |

About 2½ % of concentrates were separated from the tailings by Frue vanners and sent away for chlorination. The re-

mainder was automatically classified, giving 73·015 % sands and 25·525 % slimes for treatment, while 1·459 % slimes escaped.

At the Henry Nourse mill, the largest and most advanced on the Rand, the tailings wheel is 60 ft. diam. The tailings are classified by pointed troughs, the concentrates and coarse sands going to vats reserved for them, while the remaining sands are separated from the slimes by pointed boxes, the slimes going at once to the settling pond and the sand being distributed through 3½-in. hose to the vats. These are 12 in number, arranged on the double-treatment system, each having a capacity of 400 t. The 6 upper ones are 40 ft. diam. and 8 ft. deep; the 6 lower, 37 × 10 ft. They are made of ¼-in. steel, and are supported by steel girders and tubular steel pillars. An asphalt floor is laid under the vats and around the tailings wheel to collect leakages. Each vat has 7 bottom-discharge doors. It is expected that the troughs will yield 25 % of the tailings as concentrates with 15 dwt. gold per ton, and 4 vats are intended for the treatment of this product. The remainder of the tailings, minus a large percentage of the slimes, will be subjected to 6 days' treatment in the upper tanks, and then be dropped into the lower tanks and treated for 6 days with stronger solution.

The Jumpers mill introduced double treatment in 1899, with a result of additional extraction equal to 8·736*d.* per ton at an increased cost of 5·825*d.* per ton, leaving an augmented profit of 2·911*d.* per ton or 1670*l.* for the year.

May Consolidated uses a plunger-pump (15-in., double cylinder) for raising the tailings 55 ft. There are 17 200-t. tanks in two tiers on wooden trestles 20 ft. high. Distributors are used in charging and mechanical haulage in discharging. Average assay is 4½ dwt. Solutions employed are ·01 to ·02 % for preliminary, ·1 % for strong, and ·025 % for weak. Residues carry about ·8 dwt., KCy consumption is ·3 lb. per ton, and working costs are about 2*s.* 6*d.* Siemens-Halske precipitation is in favour, giving bullion 890 fine.

The 3¼-dwt. pulp of the Metropolitan is raised by a wheel, and classified into 25 % coarse sand (5 dwt.), 45 % medium

sand (3 dwt.) and 30 % slimes ($1\frac{1}{4}$ dwt.). Electric rope haulage is used. Treatment endures 5 days with ·08 % liquor, residues assaying $\frac{1}{2}$ to 1 dwt. Precipitation is by Siemens-Halske process. Total working cost is about 2*s.* 10*d.* per ton. Zinc has since been substituted.

New Crœsus has a 45-ft. wheel, 5 collecting vats, 5 leaching vats for sands and 2 for concentrates, pointed-trough classifiers being employed. Double treatment and Siemens-Halske precipitation are adopted.

At New Heriot the tailings are raised by a wheel, and classified by boxes into concentrates (33 dwt.), sands (5 dwt.), and slimes ($3\frac{1}{3}$ dwt.). Concentrates receive 13 days' treatment, and yield $87\frac{1}{2}$ % extraction ; sands, 5 days, and $69\frac{1}{2}$ % ; slimes are stored. Sand residues assay $1\frac{1}{2}$ dwt. Working costs are 4*s.* $2\frac{1}{4}$*d.* per ton treated ; consumption of cyanide is ·84 lb., of zinc ·3 lb., and of caustic soda ·2 lb. per ton.

The New Primrose is one of the largest Rand plants (17,000 t. a month), and has a working cost of 4*s.* 1*d.* per ton. Consumption of KCy averages ·98 lb., and of zinc and caustic soda ·22 lb. each. The treatment for some time was without classification, and the extraction only reached 55 %. Then Bettel introduced hydraulic boxes, and vacuum leaching (15 in. mercury). The eliminated 10 % of concentrates (15 dwt.) are drained, treated for 3 or 4 days with ·3 % liquor, again drained, and discharged into second treatment vats, to undergo $2\frac{1}{2}$ to 3 days' leaching, followed by water washes. Extraction is 85 to 93 %, and consumption of cyanide is about $1\frac{1}{2}$ lb. per ton. The pulp leaving the classifiers is received in distributing tank, and conveyed to intermediate tanks fitted with circular baffle plates to retain fine sands. After draining and being "vacuumed," the tank contents are levelled at surface, and a ·04 % solution is run on ; then drained ; then a stronger solution put on and allowed to stand 12 hours ; again drained under about 5 to 7 in. vacuum, and discharged to ordinary leaching tanks ; there treated with ·2 % solution, which, after standing for 12 hours, is allowed to percolate ; then with ·08 % and ·04 % solutions successively ; finishing with a water wash rendered alkaline with milk of lime. The solution from

the pumps passes through 4 filters filled with coir fibre before going to the zinc boxes, the gold mud being collected by a filter-press. The results are 75 to 85 % recovery.

A bucket wheel is used for lifting the tailings at the Robinson mill, and the sands are collected by distributors in intermediate vats, while the slimes are sold to the Rand Central Ore Reduction Co. at the rate of 2000*l.* per month for the right to treat 8000 t. of 5-dwt. grade, with proportionate increase or decrease according to rise or fall. The average fineness of their cyanide bullion is 778.

The Simmer and Jack Co. have two plants, one for sands and the other for slimes, the latter being run by the Rand Central Ore Reduction Co. The slimes overflowing from the separators at the new plant are conveyed to the old for treatment. These extensive works are probably the largest in the world, and have a capacity of about 1400 t. per day. They consist of 30 circular wooden "intermediate" vats 12 ft. deep and 24 ft. diam (200 t. capacity), 30 leaching vats 10 ft. deep and 30 ft. diam., 16 precipitation boxes (Siemens-Halske) 30 ft. long by 4 ft. 6 in. wide, 3 tailings-wheels 42 ft. diam., and pointed boxes for separating the coarser material from the slimes. The tailings are conveyed from the mill about 600 ft. to the wheels which elevate them to the separators. The slimes-overflow from the separators is carried to the old works in a launder ; the coarse material, with a small admixture of slimes, runs by gravitation to distributors, and thence to the intermediate vats. Here the material is subjected to a short cyanide treatment, and shovelled out through bottom-discharge doors (6 for each vat) into trucks which run on double tracks under each line of vats. The trucks are hauled up an inclined trestle to the leaching vats, where the tailings receive a second and longer treatment. The leaching vats are likewise provided with 6 bottom-discharge doors each, through which the residues are emptied on to a conveyor belt for transport to the waste-heap. From the storage tanks the solution is pumped to the leaching vats by 4 centrifugal pumps with 4-in. discharge. The solutions passing from the leaching vats are conducted to intermediate tanks through 1½-in. pipes.

From the intermediate tanks they are pumped up to the precipitation boxes, and thence to the storage tanks. It is estimated that this plant will operate at a cost not exceeding 2*s*. per ton.

At the Worcester works, the pulp leaving the vanners passes through hydraulic classifiers and is divided into 4 products—15 % of 15-dwt. coarse sand and pyritic matter, which is treated for 9 days with ·08 % solution and gives residues assaying 1½ to 2 dwt.; 50 % of 6-dwt. sand giving 1 to 1¼-dwt. residues after 5 days; 10 % of 4½-dwt. fine sand giving 1-dwt. residues; and 25 % of 4½-dwt. slimes not treated. The plant deals with 3000 t. a month at a cost of 3*s*. per ton, the cyanide consumption being ¼ lb. per ton.

*United States.*—A large accumulation of tailings has been worked up at the Harquahala mill, Arizona, the final portions consisting almost entirely of slimes (90 % passing 60 mesh) which demanded a special agitation plant; but their bulk did not justify the outlay, and percolation was maintained till the end, using a vacuum and treating very small quantities at a time.

Other Arizona plants are at the Congress and the Planet-Saturn mill, a preliminary roast being used at the latter Several silver mills are also in operation.

Some average figures relating to Californian practice show prices of supplies to be as follows:—cyanide, 1*s*. 9½*d*.; zinc, 6½*d*.; lime, ¾*d*.; and sulphuric acid, 2*d*. per lb.; cost of labour, 1*s*. 5*d*. to 1*s*. 10*d*. per ton; total costs, labour, supplies, etc., 4*s*. 2*d*. to 5*s*. 3*d*. per ton; consumption of supplies, ·46 lb. cyanide, ·26 lb. zinc, 4·83 lb. lime, ·17 lb. sulphuric acid per ton.

Details of the methods carried on at the Standard mill Bodie, published by T. H. Leggett,* are very interesting, as the tailings treated are of somewhat unusual character. The ore has been wet milled, passed over amalgamated copper plates, concentrated on vanners, and amalgamated in Boss continuous pans. It originally contained very little sulphurets and was comparatively low grade. In a 79 days' run, 6515 t.

* Trans. Inst. M. & M., iv. 151 (1896).

(dry) were leached at the rate of 82·4 t. daily and 2½ in. per hour. The water contents were 7 %; and as to mesh, 56 % passed through and 44 % remained on an 80 sieve. A ·2 % solution was used, and the consumption of cyanide and zinc per oz. of bullion recovered was ·6 and ·42 lb. respectively. The assay value of tailings treated was 5·208 dwt. gold and 1·96 oz. silver per ton; of residues, 1·694 dwt. gold and 1·38 oz. silver. The actual extraction was 3·4 dwt. gold (65½ %) and ·52 oz. silver (26¾ %); and the bullion fineness was 952 (712·5 gold, 239·5 silver). Supplies consumed per ton treated were :—·372 lb. KCy, ·29 lb. zinc, 7·3 lb. lime and ·13 lb. sulphuric acid.

In Colorado, the cyanidation of surface ores presents no difficulties, though the presence of tellurium oxide is often such as to materially interfere with amalgamation; but below water level the gold occurs in iron pyrites and in tellurides, necessitating either a preliminary dead roast or extremely fine grinding and prolonged treatment.

A very modern American plant is that of the Liberty Bell Co., near Telluride, Colorado, described by F. L. Bosqui.* It has a capacity of 240 t. per diem, and is enclosed in a steam-heated building to enable constant work during a rigorous winter. The wooden leaching vats are fed by launders and distributors, and are arranged in a double tier; the inside dimensions of each are 33 ft. diam. and 8 ft. deep, and the capacity is 275 t. of tailings. The upper vats are supported by steel I-beams on hollow cylindrical columns resting on stone piers. By the manipulation of overflow gates (4 to each vat), the finest slimes (amounting to 25 %) are eliminated, and the bulk retained carries about 45 % of pulp which will pass 100 mesh. The sands are discharged from the upper to the lower vats through 4 bottom-discharge doors in each; the work is done with shovels, 5 men emptying a vat in 10 hours, and the cost being 3*d.* per ton. After treatment in the lower vat, the residues are sluiced out through 4 circular bottom doors. Zinc-shaving boxes are used as precipitators. On tailings assaying 1·4 to 2 dwt. gold and 2 to 2½ oz. silver per

* En. and Min. Jl., Sept. 22, 1900.

ton, the working costs for 2 months (June–July, 1900) averaged as follows :—

| | *d.* |
|---|---|
| Labour . . . . . . . . . . . . . . | 12·250 |
| KCy, lime, zinc, and sulphuric acid . . . | 19·025 |
| Assaying . . . . . . . . . . . . | 1·825 |
| Electric light . . . . . . . . . . | ·250 |
| Shipping bullion . . . . . . . . . | ·050 |
| Taxes, insurance, and depreciation . . . | 5·900 |
| Miscellaneous . . . . . . . . . . | ·050 |
| Total . . . . . . . . . | 39·355 |

It is estimated that when working at full capacity the cost will not exceed 2*s.* 3½*d.* per ton.

At the Drumlummon works of the Montana Co., 400 t. of tailings daily are brought in by steam tramway and charged into the vats by a revolving distributor. There are 4 vats, each 38 × 9 ft., holding 400 t., and requiring 8 hours to fill. Treatment occupies 4 days. Discharge is by 2½-in. hose (2) under 60 ft. head, through 4 side and 1 bottom (central) door, occupying less than 3 hours, and costing under 1*d.* per ton. The settling tanks (4) are 22 × 14¾ ft. deep, storage tanks (2) 38 × 9 ft., and water tanks (2) 22 × 14¾ ft.; all are made of Californian redwood. Owing to the climate, operations are suspended in winter. The standard liquor is ·2 % (4 lb. per ton). At the Silver Star mill, ·05 % is used; and at other Montana plants it ranges from ·25 to ·5 % (5 to 10 lb. per ton).

In Nevada, besides many installations for dealing with old silver-mill tailings, a large plant has been erected at the De la Mar mill, Lincoln, for treating ore direct by the sodium dioxide (Kendall) process of cyanidation, with zinc fume as precipitant, and most favourable reports of its working are given.

In Utah, the Camp Floyd district almost entirely owes its development to cyanide, and was about the first field in America to adopt it. The ore is remarkable for its resemblance to silicious sinter, for carrying no silver, and for the almost invariable presence of mercury. The gold is very fine and never visible, but essentially free. In some cases the ore

gives satisfactory results in leaching at $\frac{3}{4}$ in. size, but generally $\frac{1}{4}$ to $\frac{1}{2}$ in. is best. The gold being so fine goes into solution very quickly, the time of leaching being 48 to 72 hours. Strength of cyanide solution varies from ·1 to ·2 %; ·1 % has given good extraction, but the precipitation was imperfect until the solution was fortified. Consumption of cyanide is as low as $\frac{3}{4}$ lb. per ton; of zinc, ·25 to ·375 lb. per ton of ore is used; both are governed to some extent by the judicious use of lime before leaching. With 4-dwt. ore an extraction of 75 to 80 % is realised; with 6-dwt. ore, 80 to 86 %.

Average working costs, based on coal at 20*s.* 10*d.* a ton, water at $\frac{1}{8}$*d.* per gal., KCy at 1*s.* 3*d.* a lb., quicklime at 1*s.* 8*d.* per bush., zinc at 6*d.* a lb., and wages at 10*s.* to 12*s.* 6*d.* per 8 hours' shift, are quoted by W. Magenau * as follows:—Charging, sampling, and assaying, 7·4*d.*; cyanide, 7·5*d.*; zinc, 1·1*d*; lime, ·1*d.*; fuel, 2·5*d.*; discharging (through bottom doors by shovelling into cars), 5*d.*; sundries, 1·4*d.*; total, 2*s.* 1*d.* Consumption of KCy is ·5 lb. per ton, and of zinc ·25 lb. (2·2 lb. per oz. gold recovered). Many of the smaller mills do not attempt to cleanse their bullion, but ship the gold-zinc mud at once to the smelters after having steam-dried it.

The Golden Gate works at Mercur were erected in 1898. The ore has to be raised 145 ft. on an incline 800 ft. long to the first stage in the mill, but gravitates thence. The leaching department measures 294 ft. by 60 ft., and has two floors, the main one supporting 10 rectangular tanks 50 × 25 × 5 ft., and 3 solution tanks 20 ft. diam. and 12 ft. deep, all supported on masonry piers. Charging is done by hand from trucks run on staging over the tanks. The precipitation department is 70 × 50 ft. and also two stories high. The residues are discharged into trucks and trammed to the waste-dumps. Each tank holds 260 t. and occupies 3 men a shift (8 hours) in charging. When a tank is full, strong liquor (·4 %) is admitted at bottom through 10 2-in. pipes till the mass is saturated, requiring about 8 hours; allowed to stand for 16

* En. & Min. Jl., July 21, 1900.

hours ; and followed by more of the same liquor fed in at top and drawn off at bottom, consuming 24 hours. Weak liquor (·3 %) is next applied for 2 to 3 days, and finally a water wash. "In the leaching practice followed, the aim is to have about 1 in. depth of liquor in leaching tank pass out each hour, and the best results follow when the total weight of strong and weak solutions and wash water is double the weight of ore charge ; thus in handling 700 t. of ore per diem about 1400 t. of solution are precipitated" (H. L. J. Warren *). No suction, pressure, or vacuum is used in leaching, the coarseness of the ore making a very porous charge. Discharging residues occupies 10 men 4 hours. All gold liquors and washes pass to same sump tank and are pumped up to the precipitators, 3 tanks 14 × 8 ft. deep. As the liquors enter these, they are kept in constant agitation by a ½-in. air jet under 25 lb. pressure, zinc fume being used as precipitant.

**Poisoning.**—There are at least four sources of danger from poisoning in cyanide works, viz. by drinking liquor in mistake for water ; by handling food with fouled hands ; by inhaling the hydrocyanic acid fumes given off from leaching vats, from solid residues, and from liquors ; and by inhaling arseniuretted hydrogen fumes emitted from gold-zinc mud, especially while being digested with sulphuric acid.

The first consideration is to take such precautions as will *prevent* any of these occurrences.

A supply of drinking water should be provided *outside* the mill or laboratory, contained in a filter or in a water bag securely protected against contamination of any kind ; this should be from a pure source, and should never be condensed or distilled water. It should be strictly forbidden to eat or drink anything in the mill or laboratory, and by providing a drinking supply outside, any excuse for transgression of the rule is removed. The hands and face should, of course, be washed before touching food.

Confinement of hydrocyanic acid fumes should be avoided. The greater the access of fresh air, the less the risk. Vats and

* En. & Min. Jl., Dec. 30, 1899.

tanks for leaching and storage purposes are best without any cover; if the climate renders this undesirable, the lid should not close tightly. The handling of residues, as for example in stowing them underground, is accompanied by considerable danger unless the ventilation is exceedingly good and not liable to interruption.

While collecting gold-zinc mud, an aspirator may be worn, containing a pad of cotton wool soaked in hydrogen peroxide. The treatment of the mud with sulphuric acid for dissolving the excess of zinc should always be conducted in a closed chamber or "stink-cupboard," equipped with a capacious ventilator leading well outside into the open air (away from windows), and supplemented, if necessary, by an artificial current generated by a jet of steam, compressed air, or flame.

If these preventive measures are duly observed, no poisoning can occur, but it is well also to have a remedy in constant readiness.

By far the most useful is hydrogen peroxide. It is to be borne in mind that a cyanide poison taken by way of the stomach is very rapidly decomposed by the acidity there encountered, and passes at once into the blood, so that any antidote aiming at counteraction in the stomach itself is pretty sure to fail. Less reliance, therefore, can be placed on any remedy to be administered by way of a drink, such as solution of cobalt nitrate, or the "iron cure" (10 gr. of ferrous sulphate with 1 dr. of ferric chloride in 1 oz. of water, followed by 20 gr. potassium carbonate in 1 oz. of water). In the case of inhaled fumes, of course the poison enters the blood even more directly by way of the lungs. Hence the only form of administration which can be counted upon is a subcutaneous injection. Dr. J. Loevy considers that safety can be ensured in the following manner:—

"A certain quantity of pure peroxide of hydrogen, distilled water, and a Pravarz syringe should be kept in every cyanide plant, and all workmen on the plant should be made acquainted with the method of preparing a solution of the required strength, and of injecting it subcutaneously — a

practice which any sensible man can readily acquire. The peroxide of hydrogen should be kept in small bottles, containing about ½ oz., with tight-fitting glass stopper, and placed in the dark to prevent decomposition. Considering that provisions have been made in all large industrial establishments to render first surgical aid in case of accident, I think that a provision as suggested above against cyanide poisoning is just as well called for in cyanide plants."

Prof. Kobert, who was the first to suggest this remedy, recommends a 2 % solution.

# CHAPTER XI.

## MELTING BULLION.

THE final stage through which all bullion passes before leaving the mill, whether it has been obtained in a concrete form by retorting amalgam, or in a pulverulent condition by precipitation from chlorine, bromine, or cyanide solution, is that of being melted and poured into moulds.

This process is most fittingly carried out in the assay office, which is provided with a special furnace for accommodating the "pots" or crucibles, conveniently located beside the amalgam retort and the assay furnaces, and communicating with the same chimney.

The gold sponge or mud, as the case may be, properly prepared, is filled into crucibles along with suitable fluxes, heated to fluidity, skimmed, and poured into moulds, the "bricks" being then weighed and stamped.

*Furnace.*—The type of furnace almost universally adopted is a pot furnace, as shown in Fig. 188, of dimensions suitable to the size of pot required by the amount of bullion afforded by each clean-up. The pot or crucible *a* stands surrounded and supported by coke in the fire-chamber *k*, the fire-bars *b* resting upon a couple of bearer-rods built into the masonry. Fire-brick *c* is employed as a lining to the fire-chamber *k*, horizontal flue *l*, and vertical flue *m*, to resist the intense heat of the furnace ; while common brick *d* is sufficiently enduring for the remainder of the structure, and in fact the solid portion *n*, acting merely as a base for the vertical flue *m*, may be of rubble. The front of the furnace, which is always the weakest part, being least protected against bulging, is provided with a cast-iron frame *e*, which is tied in by stout iron rods passing right through the brickwork of the furnace and the

wall of the assay office against which it is built, a shield-plate and coarse-threaded nut enabling the rod to exert considerable force in keeping *e* in place. Sometimes the cast-iron frame is replaced by straps of 6 × ¼-in. flat iron running along the front, at the level of the tie rods; but the bond is

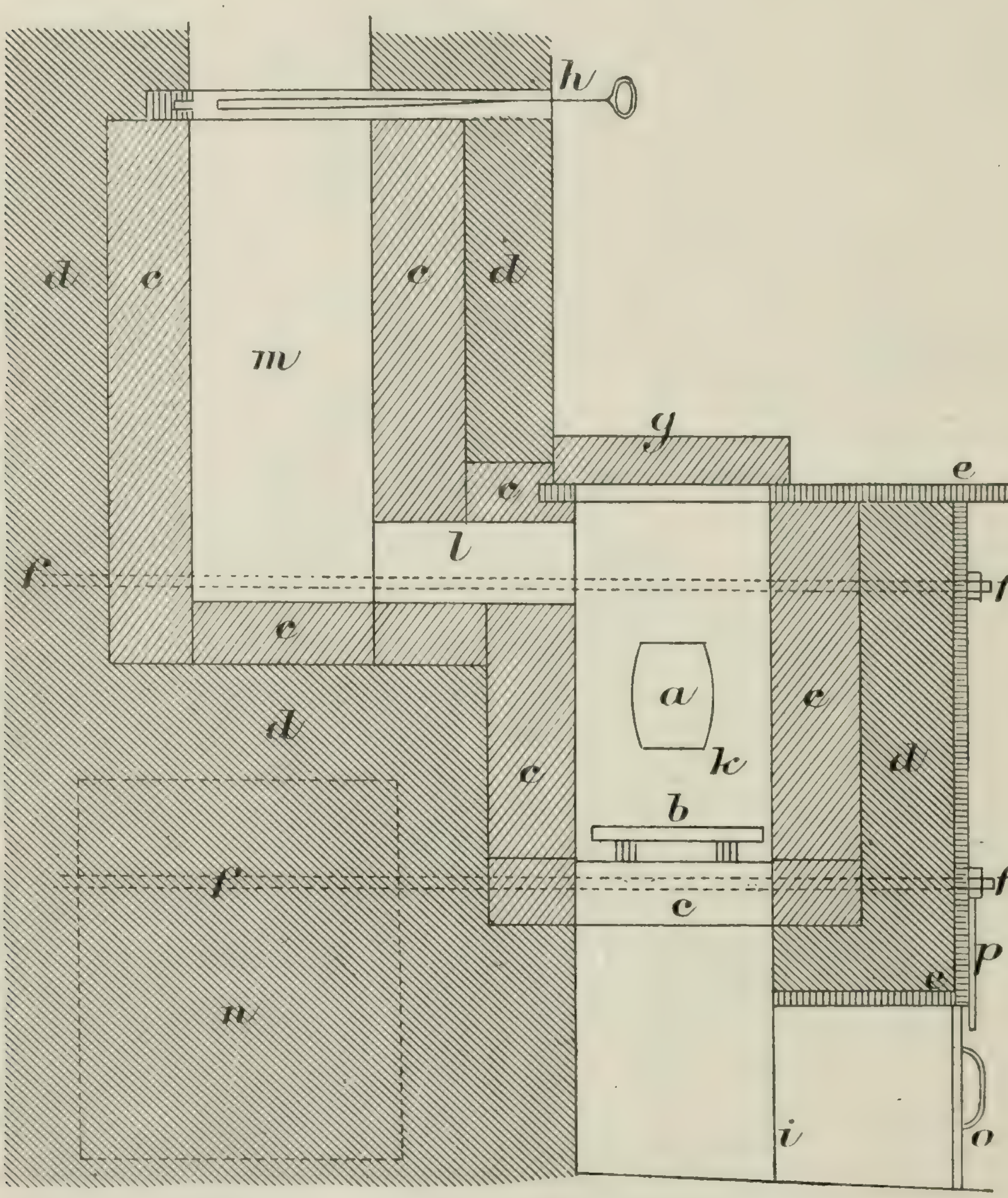



FIG. 188.—BULLION MELTING FURNACE.

not so effective. The lid of the fire-chamber is a movable fireclay slab *g*. A damper *h* is provided in the flue *m* before the latter becomes merged in the general flue leading to the stack. The ash-pit *i* is controlled by a sheet-iron sliding door *o*, held in position by a small dependent sheet-iron strap *p*; the floor of the ash-pit should slope outwards slightly, to

facilitate cleaning, and it is always well to provide a stout sheet-iron tray of the full area of the pit, to catch escaping bullion in case of any being spurted out of the pot or any accident to the pot itself.

The most suitable fuel is coke, and, as in the case of the assay furnaces, it pays to get the best quality obtainable. Failing coke, wood charcoal is occasionally relied on, but it is not so satisfactory; the hardest kinds are best, and all must be freed of small particles, say $\frac{3}{4}$ in. and under. When using charcoal, the pot must stand on a slab of fireclay or a fire-brick placed on the bars, and often forced draught from a bellows or compressor is necessary to produce sufficient heat. Anthracite coal is favoured by some operators.

*Crucibles.*—The crucibles or pots used for melting bullion are of three kinds—plumbago (graphite), a superior sort of graphite mixture known as "salamander," and clay.

The graphite or salamander crucible is most costly, but when the bullion is sufficiently pure to need practically no fluxes, its endurance (generally surviving about 8 fusions) makes it come cheapest in the end.

The ordinary clay pot of a size adapted to melting any considerable quantity of bullion is not reliably strong, though it finds favour in some quarters when much fluxing is to be done. Nevertheless, at many mills, the graphite article is adhered to even when using fluxes, notwithstanding its shortened life and greater cost.

Preliminary annealing, though not essential to salamander pots, is absolutely so for all others, and is advisable in any case. A very good plan, where facility offers, is to keep the pots stacked mouth downwards on a flue, or over an assay furnace, or in any position where they will be maintained hot and dry. But even this will not obviate the need of a special heating just before use, commencing with a very gentle fire, and finishing by rendering the pot red-hot. This may best be done in the pot furnace itself.

Means of lifting out the red-hot pot and pouring its liquid contents must be provided. Usually they take the form of home-made tongs, various patterns being favoured. Ordinary

tongs for nipping the wall of the crucible can only be used for a direct upward lift, as for instance in disengaging the pot from its bed of coke ; if they are allowed to exert any lateral strain on the wall of the pot, as for pouring, they are pretty sure to break a piece out. So-called "basket" tongs (*a*, Fig. 189), embracing the whole pot, are very useful ; and a single

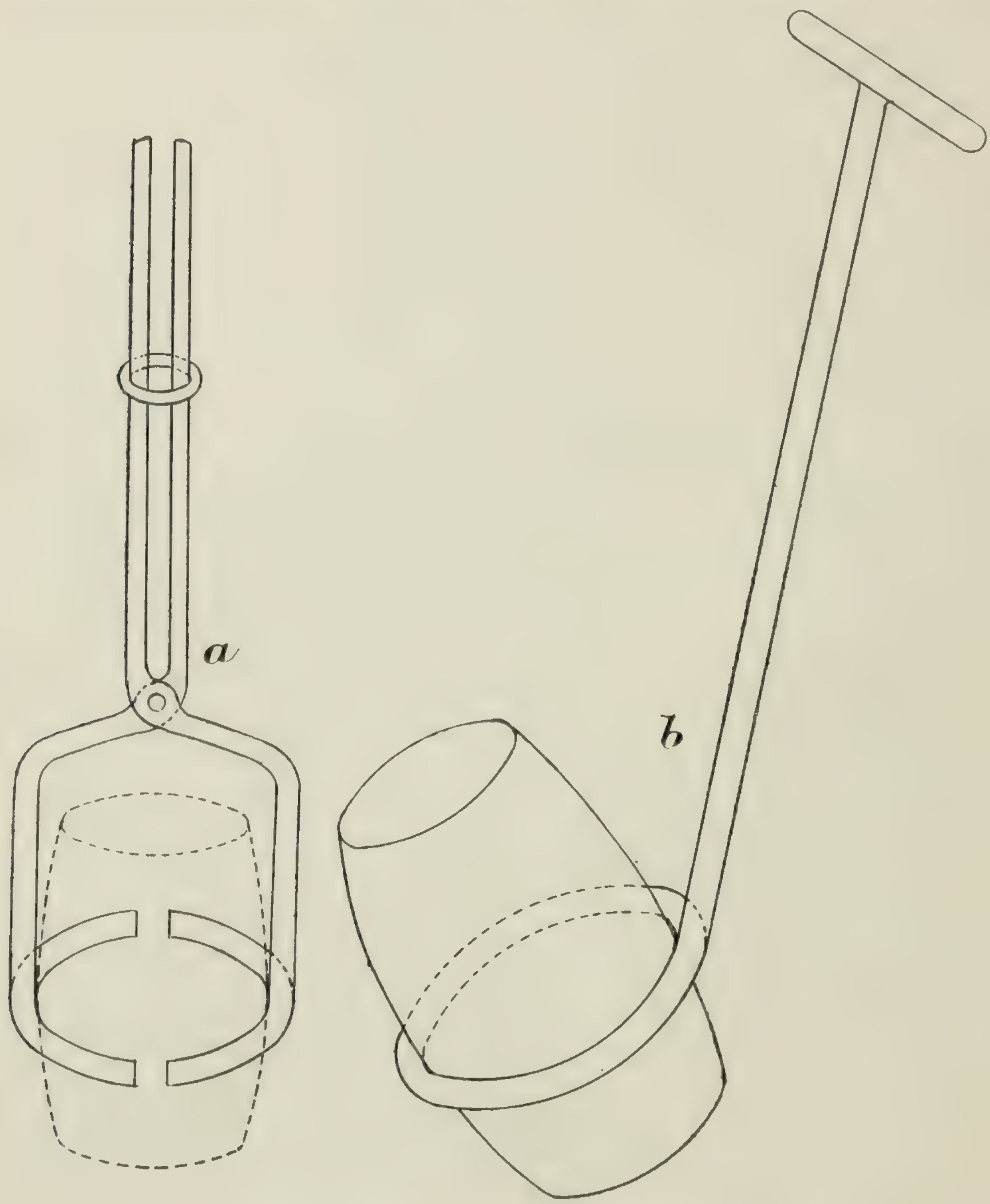



FIG. 189.—BULLION POT TONGS.

solid ring *b*, with firm looped or cross-barred handle, is convenient for pouring. Where melts of 500 oz. and upwards are general, the pot should be swung out and tipped by the aid of a small crane or light chain and pulley, as the task of lifting and pouring by manual effort alone is very severe, and rendered still more so by the exposure to most intense heat.

*Fluxes.*—The character of the gold to be melted varies between very wide limits. Thus, chlorination gold is remarkably pure when thrown down by iron sulphate or sulphuretted hydrogen, and needs but little fluxing, while when charcoal is used for precipitation the bullion is bulky and dirty from contamination with ash derived from the charcoal. Sponge gold from amalgam often contains iron and copper from the battery, and lead, zinc, and antimony from the ore. But worst of all is cyanide bullion, which, without previous acid treatment, is much fouled with various base metals, the most conspicuous being zinc and lead, as well as a certain amount of quartzose matter.

With really clean bullion, very little flux is necessary, though a sprinkle of borax is almost invariably used. Sometimes salt is preferred, though the risk of loss of gold by volatilisation must in that case be very great.

For foul bullion, a proportion of soda bicarbonate is a necessary addition.

Much divergence is noticeable in the selection of fluxes made by different operators, as the following examples show :—

FLUXING MIXTURES.

| | Bullion. | Borax. | Soda. | Nitre. | Sand. | Fluorspar. | Remarks. |
|---|---|---|---|---|---|---|---|
| *a* | 300 | 80 | 150 | .. | 50 | .. | Clean KCy mud. |
| *b* | 300 | 120 | 150 | .. | 50 | .. | Zincy ,, |
| *c* | 300 | 100 | 200 | .. | .. | 20 | Sandy ,, |
| *d* | 300 | 20 | 40 | .. | 20 | .. | Clean ,, |
| *e* | 300 | 150 | 90 | .. | .. | .. | ,, ,, |
| *f* | 300 | 120 | .. | .. | 75 | .. | ,, ,, |
| *g* | 300 | 150 | 30 | .. | .. | .. | ,, ,, |
| *h* | 300 | 150 | 60 | 30 | 15 | .. | ,, ,, |

The mixtures *a*, *b*, *c*, *g*, and *h* are taken from Transvaal practice, and *d*, *e*, and *f* from Californian : *f* is said to form a very liquid slag from which the gold settles very readily ; it does not "boil up," and repeated additions can be made to

the charge as it melts down. Soda needs to be increased when much sand is present, until it amounts to $2\frac{1}{2}$ times the weight of the latter ; fluorspar supplements the soda. For metallic impurities, the borax must be augmented, and old slags may be usefully added.

When lead acetate is used for coating the zinc for cyanide precipitation (see p. 613), much lead finds its way into the gold mud. In one instance, T. L. Carter * found 23 %, but by employing a clay liner to the graphite pot, and a flux of 60 % borax, 19 % nitre, $11\frac{1}{2}$ % sand, and 7 % soda, he obtained a fineness of 876.

In melting the gold dust derived from blanket concentrations and arrastras, F. Owen † applied one-third its weight of iron sulphide, as clean as possible, preferably slags from previous fusions ; when the crucible contents were well melted, scrap iron was carefully added bit by bit to effect desulphurisation.

The proper cleansing of cyanide gold-mud — dissolving away zinc by sulphuric acid, and washing charcoal gold-ashes with warm water to remove soluble salts—before it reaches the pot, effects a great saving in both fluxes and crucibles.

Of the fluxes in use, borax is most important and most costly. Moreover, the ordinary commercial article contains nearly 50 % of water, and many of the so-called "calcined" brands are but little better ; this creates a violent ebullition and swelling in the crucible, and may easily cause material loss of gold by spurting. The special ground borax glass furnished by the Cyanide Plant Supply Co., London, is very much superior in quality, and ultimately more economical, as less is used, and larger charges are possible in the crucibles.

*Charging.*—The crucible is made red-hot before any portion of the charge is admitted.

With amalgam bullion, the sponge is merely broken up (by hammer and cold chisel if necessary) into pieces of convenient size, and introduced by assay tongs or a scoop, a little borax being finally added. If the sponge has come

* Jl. Chem. & Met. Soc. S. Africa, Oct. 1898.

† Trans. Inst. M. & M., iv. 19 (1895).

directly from the retort, it cannot be anything else than dry, but if from a safe or other store, it must be first well dried on the top of the furnace.

Gold mud or ash, being in a powdery condition, is very easily spilled and scattered by even a strong air current, and for this reason it is best passed into the crucible by the medium of a sheet-iron tube with funnel mouth. The tube should be so long that it will reach right into the crucible without causing the operator to stand uncomfortably near the furnace. The introduction must be done with the least possible force, so as to avoid disturbing any existing contents. But there are bound to be mechanical losses caused by pouring in an impalpable powder and exposing it to the energetic air currents circulating around a red-hot crucible. It is therefore much better to aggregate the gold dust by forcible compression, using a small hydraulic or other press for making the cakes, when a scoop suffices for placing them in the pot. As melting proceeds, the bulk shrinks, and further additions can be made to the charge, a proportion of flux following each quota of bullion.

At some mills, precipitated (dusty) gold for melting is mixed thoroughly with the fluxing material before admission to the crucible; and at Mount Morgan it is the custom to incorporate 2 parts gold mud, 1 of borax, and 1 of soda ash (instead of bicarbonate), dry, on a concrete floor, and then to moisten the mass with water, in which way is obtained a combination which is immediately formed into a hard cake on encountering heat, and from which no loss in dust can take place.

No crucible should at any time be filled more than three-fourths full; in case of a tendency to boil over, a little salt must be quickly added.

*Slags.*—The slag resulting from a melt varies in volume and composition as the bullion and fluxes vary. It always contains some small proportion of gold, and, with cyanide bullion, a large amount of zinc and no little lead, according to the precipitant employed. The least valuable slag commonly assays 1 oz. per ton, even when the bullion turned out is 960 to

980 fine. A very usual estimate is that it will carry $\frac{1}{4}$ to $\frac{1}{2}$ % of the bullion output. At the Standard mill, Bodie, Calif., R. G. Brown * got 37·6 lb. slag for each 1000 oz. KCy gold mud melted or 481 oz. gold bullion produced. In this slag, the ratio of gold to silver was 1 to 6·5 while in the bullion it was 1 to 2·9. In 1896, the Standard mill produced 1325 lb. slag, having a value of 2*s.* 2*d.* per lb., equivalent to 33·1 lb. or 71*s.* 8*d.* per 1000 oz. gold mud melted; and at the Treasury mill, the figure was 31·1 lb. In much of the Transvaal slag, A. C. Claudet † finds 25% zinc and upwards, some of it occurring in beads or buttons. Such slag is very infusible, and the bullion does not readily settle out of it, in consequence of which it carries a very high bullion contents, Claudet reporting from 40 oz. to as much as 400 oz. per long ton.

This certainly seems a ridiculously high assay for a slag, and closely approaches that of base bullion. It is the more serious when it is remembered that such gold is not recoverable at the mill by simple grinding and amalgamation. It has to be shipped to the lead and copper smelters. Hence the maximum of concentration is desirable. With ordinary slags containing say 1 oz. bullion per ton, passing them through a battery or an amalgamating barrel or pan is often considered sufficient, and inasmuch as the tailings from this operation are in all likelihood again submitted to cyanidation, probably a merely nominal loss is suffered. But with rich slags the case is different.

An ingenious way of concentrating the bullion in the case of rich slags has been suggested and carried out by Caldecott ‡ in S. Africa. The slag is allowed to accumulate till a sufficient quantity is obtained, say, 1 cwt. It is then re-melted and cast into a conical iron mould, 2 in. from the bottom of which a $\frac{1}{2}$-in. hole is bored, which is temporarily plugged with fireclay. About one minute after casting the slag, an iron rod is thrust through the hole, and breaking through the skin of frozen slag on the side of the mould, forms a passage through which the clean and still molten slag in the centre can flow

* Trans. Inst. M. & M., iv. 251 (1896). † *Ibid.*

‡ Jl. Soc. Chem. Ind., xvi. 503 (1897).

away. The crust remaining will be found to contain, in a concentrated form, all the gold originally present in the slag. These crusts accumulate, and are treated similarly, until a very small amount of slag very rich in gold is obtained, which is readily melted for its valuable contents. The slags finally rejected assay less than ½ oz. per ton, and can be still treated in the battery.

*Treating Voluminous Gold Ash.*—The enormous output of gold-bearing ashes (from incineration of charcoal) at Mount Morgan, reaching 30 cwt. a month, or 60 cwt. with fluxes added (1 borax and 1 soda ash to 2 gold ash) renders melting in crucibles very laborious and costly, besides which the slags are always full of gold beads. Consequently the reduction officer * designed a small reverberatory, like an ordinary steel furnace, with a cast-iron pan to hold the brick hearth. The pan is supported on 4 screw standards which can be moved up and down and the whole hearth withdrawn from the furnace when necessary. The pan is lined with fire-brick. He tried smelting in a sand-hearth, but gave it up in favour of the simple fire-brick hearth. After 6 months' work with the furnace, during which time 75,000 oz. gold were melted, 25 ft. of the flue torn down and crushed in a battery yielded only 35 oz. gold, most of which came from the first 8 ft. of the flue. The fire-brick hearth soon absorbed some gold, as the joints were destroyed by the alkaline fluxes ; but once in 3 or 4 months, the hearth was torn out of the cast-iron pan, and the accumulated gold beads were recovered, some by simple picking and the rest in a battery. Even a new hearth every month would compare very favourably in cost with the crucibles that would otherwise be destroyed. At first much pains was taken to get a melting heat through the 6-in. layer of slag above the gold ; but it proved to be unnecessary, the gold becoming semi-fluid in 4 to 5 hours, and settling down out of the slag in irregular lumps on the hearth. The slag was then tapped out into a large mould that would hold 9000 oz. bullion (in case of any gold coming), and thence into slag pots ; it contained 20 to 30 oz. non-amalgamable gold per ton. Next the furnace was re-

* En. & Min. Jl., Dec. 17, 1898.

plugged and fired heavily for $1\frac{1}{2}$ hour, when the whole of the gold was brought to fusion, and could easily be tapped out into a mould. Re-melting in a crucible followed.

*Cleaning and Pouring Molten Gold.*—The common practice in cleansing the fused gold in the pot is to repeatedly skim the surface with an iron tool, alternately renewing additions of flux, until the metal shows a greenish yellow tint, with a clear bright surface. But Carter finds that a small jet of air, from a compressor, playing upon the molten metal, is most effective in oxidising base metals, especially lead, that being a notable impurity when leady zinc is used in KCy precipitation. Other operators introduce additional borax glass with some nitre and bi-chloride of mercury at the last stage of the fusion, and pour bullion and slag together. Again, it is recommended to employ a clay-lined graphite pot, using more nitre and some sand and bone-ash. Copper, derived from amalgamating plates, may be removed by adding sulphur, but that will entail a silver loss. Copper is sometimes purposely added, to expel antimony, arsenic, and bismuth, and toughen the metal. The appearance and ductility of bullion may be improved by granulating (pouring it into water), giving it a sweating roast in a muffle, and re-melting.

Moulds for receiving the poured metal are made of cast iron, in various shapes and sizes, sometimes bearing the initials of the mill owners (reversed), so as to save stamping the bricks. They are made absolutely dry, and then sometimes the surface is greased, or some vegetable oil is poured into them, the fatty matter giving a good surface. But when fluxes are used in melting, no grease is admissible, and the poured metal is covered with various coatings, salt, sugar, borax, soda, and nitre, all having their advocates. While still quite hot, the bricks are turned out of the moulds, and plunged into a bath of dilute nitric or sulphuric acid. After scrubbing and drying, they are weighed, and packed in a steel box for conveyance to bank or mint.

*Cost of Melting.*—Much depends on the source of the bullion, its initial purity, and the amount of purifying necessary, in determining the cost of melting. With amalgam bullion, it is usually a very small matter. In many of the

cyanide mills of the Transvaal and New Zealand, it is computed at 2*d.* per oz. from gold-zinc mud, the resulting bullion being not more than 800 fine. On charcoal-precipitated bullion at the South German cyanide works, Victoria, it reaches 1*s.* per oz. exclusive of labour, the fineness being 900; but in this case there is an extravagant waste of material through not first washing out soluble alkaline salts.

*Bullion losses.*—The loss in weight sustained by the sponge or mud in the process of melting will depend on the purity of the article melted. This, for by far the greater part, is not a loss of bullion at all, but of impurities accompanying it, such as mercury, lead, zinc, iron, and other base metals, together with some silicious matter, either passing into the slag, or being volatilised. Such a diminution often amounts to 5 to 10 %, and in exceptional cases reaches 25 %.

At the Lucknow mill, New South Wales, in 4 years 1896–9, on 23,725 oz. gold sponge from amalgamation the loss of weight amounted to 13·57 %. But in this case there was always an appreciable amount of antimony in the sponge, and it was not entirely removed by melting. A very curious fact was noticed: when antimony was more abundant than usual the ultimate bullion was always of higher standard, as if the volatile metal had displaced a portion of the silver alloyed with the gold. The average fineness of the total bullion was 867 gold and 133 silver; but while the amalgam bullion returned only 856 gold and 144 silver, the concentrates, which carried the bulk of the antimony, assayed 885 gold and 115 silver.

Some slight actual loss of bullion nearly always occurs in melting, by metal enveloped in the slag, spurted out among the fuel, and adhering to the pot; but all this should be ultimately recovered by passing the old pots, slags, ashes, etc., through the battery.

Under certain circumstances, a much more serious loss may take place by volatilisation, though whether the gold itself is vaporised or whether it is mechanically entrapped among zinc and other volatile fumes is not clear. R. M. Aitken * has described an instance where he collected over

* En. & Min. Jl., Dec. 31, 1898.

1000*l.* worth of gold from the flue dust of a New Zealand cyanide mill melting plant, which is a very emphatic proof that such losses must be carefully guarded against.

*Fineness of Bullion.*—In the matter of "fineness" or ratio of precious metals, chlorination bullion easily comes first, that process being in itself a kind of purifying method, even to the extent of eliminating silver. Absolutely pure gold is stated at 1000 fine, and chlorination bullion ranges from about 985 to very nearly the full figure.

Amalgamation bullion always embraces more or less silver, but with that exception it is generally of good standard. Ordinary fineness runs from 750 to 850 gold, and from 145 to 245 silver, the remaining 5 points being chiefly made up of minute quantities of iron and copper. On over 25,000 oz. bullion extracted at Lucknow, the loss in refining was 1·73 %, and the refined bullion assay was 856 gold and 144 silver. In this case, antimony and arsenic contributed to the impurities. Some South African examples are—Angelo, 877·5; Driefontein, 866·3; Jumpers, 875·56; New Comet, 888·6.

Cyanide bullion varies very much, according to the amount of care taken in removing base metal before melting. This base metal is chiefly zinc (and sometimes lead) from the precipitation process. A few examples from current practice may be quoted. Transvaal cyanide bullion, according to A. C. Claudet,* who has assayed many tons of it, approximates the following composition—643 gold, 67 silver, 150 zinc, 118 lead, and 10 copper, which leaves 12 points unaccounted for, probably represented by iron. The Crown Deep bullion is 850 gold, 90 silver, 40 copper, and 15 iron. At Luipaard's Vlei (S. Africa) trouble is taken to purify the bullion, and its total fineness (gold and silver) is 986; and at the Standard mill (California), the combined fineness is 952. The Myalls (New South Wales) bullion is only worth about 50*s.* an oz., equal to 600 fine. Much of the New Zealand bullion falls far below this even. The cleanest bullion produced by the Mysore Co., India, assays 81·3% gold, 6·9 silver, 2·71 lead, 6·78 copper, ·4 zinc, and ·12% nickel.

* Trans. Inst. M. & M., iv. 252 (1896).

*Refining Bullion.*—While the absolute refining of bullion is an operation which must always be left to the mint or other refiner, the removal of excess of impurities, or better still, their elimination before fusion of the bullion is made at all, can always be undertaken at the mill. When it is borne in mind that refiners charge 4*d.* per oz. for refining base bullion, it will be evident that it is worth while to avoid impurities as far as possible.

This has led to a suggestion that plumbified zinc (p. 613) shall give way to a pure brand of lead-free zinc for precipitation. Alfred James * insists that the adoption of such a precipitant will "avoid the refiners' charges of, on an average, 9*d.* per oz. (of fine gold); will lessen the amount of mud to be treated, and the loss during clean-up of ·2 %. Against this is the greater price of lead-free zinc, and as it takes, with low-grade tailings, an average of 1 lb. zinc to produce 1 oz. gold, it is easy to calculate that, even if lead-free zinc were 11*d.* per lb. dearer than ordinary zinc, it would still be preferable and cheaper to use the former brand; but, as a matter of fact, the difference in price is very much less than this sum," in reality about $\frac{3}{4}$*d.* per lb. more for zinc containing ·15 % lead than that with $1\frac{1}{2}$ to $2\frac{1}{2}$ %. Inasmuch, however, as zinc can be removed by sulphuric acid before melting, and lead by an oxidising jet during that process, it would seem that the reported benefits of leady zinc can be retained without necessary detriment to the bullion.

The difficulties of correctly sampling and assaying base bullion have been urged at some length by A. C. Claudet † as inducements to avoid it; and he has worked out a number of examples showing what actual monetary losses it entails on the seller, taking as a basis that 850 oz. fine gold will give 1000 oz. of good bullion.

Some cyanide bullion is roughly 800 fine, although others go as low as 300 fine, the latter containing an enormous amount of impurities, which render the task of correct assaying extremely difficult. On the above basis, cyanide bullion

* Trans. Inst. M. & M., vi. 8 (1897).

† *Ibid.*, iv. 252 (1896).

BULLION: CHARGES AND VALUES ACCORDING TO FINENESS. (A. C. Claudet.)

| Fineness per 1000. | Weight of Bullion. | No. of Bars. | No. of Assays. | Melting per oz. Bullion. | Refining per oz. Bullion. | Per oz. Standard. | Value of Bullion. | | | Melting, Assaying, Refining. | | | Nett value of Bullion. | | | Value realised per oz. Fine Gold. | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | oz. | | | *d.* | *d* | Premium *d.* | Plus premium £ | *s.* | *d.* | £ | *s.* | *d.* | £ | *s.* | *d.* | *s.* | *d.* |
| 850 | 1000 | 2 | 2 | $\frac{1}{4}$ | 4 | $\frac{1}{2}$ | 3606 | 16 | 1 | 18 | 2 | 2 | 3588 | 13 | 11 | 84 | 5 |
| | | | | | | Deduction per mil. | less deduction | | | | | | | | | | |
| 800 | 1062 | 2 | 4 | ,, | ,, | 2 | 3595 | 17 | 3 | 19 | 11 | 0 | 3576 | 6 | 3 | 84 | 2 |
| 700 | 1214 | 2 | 4 | ,, | ,, | 3 | 3589 | 8 | 6 | 22 | 5 | 11 | 3567 | 2 | 7 | 83 | 11 |
| 600 | 1417 | 3 | 6 | ,, | ,, | 4 | 3580 | 16 | 10 | 26 | 5 | 10 | 3554 | 11 | 0 | 83 | 8 |
| 500 | 1700 | 3 | 6 | ,, | ,, | 5 | 3568 | 16 | 6 | 31 | 6 | 1 | 3537 | 10 | 5 | 83 | 3 |
| 400 | 2125 | 4 | 8 | ,, | ,, | 5 | 3568 | 16 | 6 | 39 | 4 | 7 | 3529 | 11 | 11 | 83 | 1 |
| 300 | 2833 | 5 | 10 | ,, | ,, | 5 | 3568 | 16 | 6 | 52 | 3 | 4 | 3516 | 13 | 2 | 82 | 9 |

Each bar contains 927·3 oz. standard gold at 77*s.* 9*d.* = 3604*l.* 17*s.* 6*d.*

of 800 fine gold would produce 1062 oz.; total bullion 700 fine, 1214 oz.; 600 fine, 1417 oz.; 500 fine, 1700 oz.; 400 fine, 2125 oz.; and 300 fine, 2833 oz. As the fineness lessens so the weight of bullion increases, thereby making the costs higher. With regard to expenses, owing to the difficulty of ascertaining what is really the fineness of cyanide gold, the practice is to have two "dips" and two assayers; one assays one "dip," and the other assays the second "dip," and the mean of the two assays is taken as the value of the bar (500 oz.) This increases the number of assays, and, consequently, the lower the fineness, and the more impurities, the greater the number of assays required. The cost of melting, no matter whether the gold be fine or coarse, is $\frac{1}{4}$*d.* per oz. of bullion—a charge which, although not appearing large in itself, is, on a heavy output, a matter to be considered. The charge for refining is 4*d.* per oz. The price paid for ordinary good parting bullion (gold and silver) is $\frac{1}{2}$*d.* premium, *i.e.*, if the Bank price is 77*s.* 9*d.* per oz. standard, the price paid is 77*s.* 9$\frac{1}{2}$*d.* In the case of cyanide bullion of 800 to 700 fine, 2 per mil is deducted from the assay; 700 to 600 fine, 3 per mil; 600 fine, 4 per mil; and 500 to 300 fine, 5 per mil. The price of gold is ascertained by finding out the number of fine oz. in a bar, and converting it into standard gold at the rate of 77*s.* 9*d.* (77*s.* 10$\frac{1}{2}$*d.* in some of the Colonial Mints). If one has gold muds furnishing bars in one case 800 and in the other 500 fine, the deduction in the former case will be 9*l.*; in the latter, it will amount to 36*l.* Refining charges on 800 fine will be 17*l.* 14*s.*; on 300 fine, they will amount to 47*l.* Melting charges on 800 fine will be 1*l.* 1*s.*; on 500 fine, 1*l.* 15*s.* 5*d.* Assay charges on 800 fine will be 16*s.*; on 500 fine, 1*l.* 4*s.*; on 300 fine, 2*l.* The net values realised will work out as follows: Ordinary battery gold, 850 fine, will fetch 84*s.* 5*d.* per oz. fine gold; cyanide gold, 800 fine, 84*s.* 2*d.*; 700 fine, 83*s.* 11*d.*; 600 fine, 83*s.* 8*d.*; 500 fine, 83*s.* 3*d.*; 400 fine, 83*s.* 1*d.*; 300 fine, 82*s.* 9*d.* The silver in the bars is paid for in full at fine silver price, and does not enter into the figures given.

The table opposite prepared by Claudet makes the matter still more clear.

# CHAPTER XII.

## SMELTING ORES AND CONCENTRATES.

By smelting is understood a kind of concentration by means of fire, advantage being taken of the presence of fusible and non-volatile base metals to collect the precious metals.

There are two distinct types of such smelting. In the one, known as "matte smelting" or "pyritic smelting," use is made of a portion of the sulphur in the ore to support combustion and economise fuel, the product being a "matte" of metal and residual sulphur; while in the other class, all sulphur is removed from the ore by a preliminary roasting operation, and heat is obtained from added fuel alone, the product being a more or less pure metal. In the first class, iron forms the bulk of the metal contained in the matte; in the second, copper or lead is predominant.

A description has already (p. 316) been given of the methods employed for isolating the sulphides or "mineral" from battery pulp after the free gold has been recovered by amalgamation. In many cases, these concentrates are then subjected to roasting (p. 361) and chlorination (p. 410), and in fewer cases to cyanidation (p. 473), with or without previous roasting. Under some circumstances, however, neither of these methods commends itself, and recourse is then had to smelting, either to matte or to metal.

Smelting is very rarely conducted on a mine, the concentrates being sold as such to the smelting works. These are often very far distant from the mill where the concentrates are produced, so that cost of transport becomes an important item, necessitating the utmost possible reduction in bulk before shipping. But "close concentration" cannot be effected without loss of value in the tailings; and the best course to be

followed must depend in some measure upon what the ultimate treatment of the tailings is to be. For chlorination, and for distant shipment to the metal smelter, the cleaner and richer the concentrates the better. For matte smelting, close concentration is not required, and may even be detrimental in a sense, as a certain proportion of gangue, especially if silicious (quartzose), is actually necessary for fluxing, and must be added in the furnace if not existent in the ore.

What has been said regarding concentrates applies equally well to certain classes of so-called "refractory" ore, which are not adapted for other treatment than smelting.

**Matte Smelting.**—Concentration of the precious metals contained in certain (as a rule very poor) auriferous pyrites by fusion has long been practised in Europe, having originated it would seem in Hungary. It is dependent on the familiar fact that when burned in a limited amount of air, iron pyrites loses about half its sulphur by combustion, and becomes mainly converted into a lower sulphide which is easily fusible.

The aim of pyritic smelting is to make the pyritic material of the ores act as fuel in a blast furnace, so that no carbonaceous fuel need be used. Ores capable of absolute pyritic smelting are very rare, and therefore in most cases the operation is carried on with the help of a slight percentage of carbonaceous fuel. The ore acts not only as a fuel but also as a collector for the precious metals, as well as for copper, cobalt, and nickel ; it also serves as a flux. The ore is consequently used raw. A preliminary roasting is not required ; only the necessary fluxes are added.

*Scope.*—The Hungarian practice of relying upon the protosulphide of iron as a collector of precious metal is, however, not to be commended, as the losses of value incurred are heavy, and it is not advisable to attempt the matte smelting of iron or arsenical pyrites in the absence of some proportion of copper or lead. But a very moderate proportion of copper pyrites will suffice for admixture with a larger quantity of noncupriferous material. The higher the copper contents, the greater the degree of concentration possible without risk of

loss. Concentration must be carried to such a point that it will pay to ship the product to the ultimate metal smelter; and this degree of concentration must be accomplished in one operation, and not by subjecting a low-grade matte to a second smelting for enrichment.

The true field of matte smelting will be found where the following conditions prevail:—

(*a*) Ores carrying value in gold, silver, and copper, principally, as well as nickel and cobalt; each being in sufficient quantity to be worth collecting, yet not enough to warrant shipment in the crude state;

(*b*) Presence of sulphides and arsenides in such proportion and condition as to preclude successful wet concentration;

(*c*) Presence of lead or zinc, and of carbonates of lime and magnesia, in quality calculated to hinder efficient roasting as well as leaching—thus preventing chlorination and cyanidation;

(*d*) Presence of silicious gangue for fluxing purposes, itself carrying a proportion of value not satisfactorily recoverable by amalgamation or concentration;

(*e*) Absence of good and cheap fuel—as for instance, coke costing 5*l.* a ton, and wood and coal in proportion—precluding direct smelting to metal;

(*f*) Costliness of labour and transport, obstructing shipment in the crude state and mechanical preparation for other treatment.

It is, of course, not to be understood that for successful competition with other extraction processes matte smelting demands the simultaneous existence of all the above conditions; but the more they prevail the more marked will be its superiority.

Matte smelting of auriferous ores and concentrates has been developed but little outside of the United States, but there it has long been most firmly established in several centres, such as Deadwood, S. Dakota; Toston, Montana; Mineral, Idaho; and Leadville, and several other places, in Colorado.

*Ores.*—The extensive works of the Deadwood and Delaware Smelting Co., long managed by Prof. Franklin Carpenter

with conspicuous success, have recently passed into the hands of the Golden Reward Co., which at one time operated exclusively by chlorination, and latterly to some extent by cyaniding. The ore supplied to the smelting works embraces three distinct kinds. One variety is a low-grade concentrate (about 5 dwt.) from the Homestake mills, saved on bumping tables, and consisting essentially of iron pyrites. With this is mixed a proportion of the silicious refractory ore from the Potsdam beds, assaying about 1½ oz., and usually treated at other works by chlorination ; this is supplied in lump form, being only cracked in a breaker. To it is also added massive pyrites of very low grade but carrying a little copper, which is so useful in collecting the precious metal, though almost valueless in itself.

The ores treated at Mineral contain practically 30 % each of quartz and calcite, with 12½ of sulphur, 10 of iron, 2½ of arsenic, and a little zinc. They are smelted in one operation, without admixture of any other ore or addition of any flux, to a very rich matte. Most of the sulphur and arsenic suffer combustion, economising fuel, while the liberated iron and zinc pass into the slag and assist concentration.

These works are controlled by Herbert Lang,* who finds the most suitable constituents for smelting to be about 30 % silica, 10 to 15 of sulphur, 10 of iron, and 30 of alkaline earths (lime, magnesia, baryta, etc.). The more sulphur, the less fuel required, arsenic assisting in the same way. A few per cent. of lead and copper are desirable for collecting purposes, and a little zinc does no harm. While baryta (heavy spar) in any quantity would preclude lead smelting, in matte smelting it may be tolerated up to even 25 %, because in the absence of coke fuel there is no reducing of the sulphates.

At the Leadville works of the Bi-metallic Smelting Co., raw ores are fed into the furnace with fluxes but no fuel, the usual contents of sulphur being about 12 %. The richest matte produced carries 33 oz. gold, 258 oz. silver, and 14·2 % copper, the return of gold and silver being respectively 3·43 % and

* Author of 'Matte Smelting,' 1899.

4·15 % below the actual amounts present in the ore. Every 9½ t. of ore yielded 1 t. of matte.

In a works at Keswick, California, Lang concentrated 4 t. cupriferous pyrites, 2 t. argentiferous limonite, and 2 t. gold quartz into 1 t. of matte.

A form of pyritic smelting is used at Mount Lyell, Tas., reducing 5 t. or more of pyrites to 1 t. of matte ; by re-smelting this matte a product is got carrying 50 % copper and all the gold and silver. The ore contains 83 % $FeS_2$, 14 % $CuFeS_2$, 2·38 oz. Ag, and ·09 oz. Au.

It is difficult to smelt fine concentrates alone, but the difficulty is much reduced by briquetting, using lime as at once a binder and a flux. Some form of press is necessary, such as is used in brick-making.

*Flux.*—Of primary importance in this as in all forms of smelting is the fluxing material necessary to secure a proper separation of metal and slag (gangue). While almost any slag that will melt at all would seem to be admissible, preference must always be given to an acid slag. Whereas in lead smelting the maximum limit of silica in the slag is about 37 %, in matte smelting it may go as high as 52 %.

Lang * has laid it down as an infallible rule that the degree of concentration depends directly upon the proportion of silica supplied. When the silica is diminished, the iron remains in combination with sulphur, and the matte fall is increased, while its copper contents are decreased. No matter how the charge is balanced, the silica controls the chemical changes to a very great extent, and the policy should be to carry as high silica as the furnace will stand in order to induce the sulphur and iron to part company. If, by reason of a change in the character of the silicious ore, the available silica is momentarily decreased, the quantity of matte is increased at once, and the copper assay runs down. On the other hand, an increase in the proportion of silica beyond what can be taken care of by the furnace produces crusts in the tuyere region, and may injure the smelting for a considerable time,

* En. & Min. Jl., July 10, 1897.

and even compel the stack to be blown out. It is in smelting ferruginous ores that this effect is noticed most, for any excess of quartz over the amount necessary to produce a singulo-silicate of iron is felt at once by the growth of crusts.

Ores containing alumina obscure the indications a great deal, and whereas a slag containing iron only as a base seems to form singulo-silicates in the pyritic stack, aluminous slags vary quite widely. The iron singulo-silicate comes down with about 30 % silica; while aluminous slags produced at the same temperature vary from 24 to 40 % silica, and the effect of the changes in the silica remains quite obscure.

As for the iron, Lang's observation is that the tendency is invariably to the formation of the singulo-silicate, and that if left to itself the more acid silicates are not formed, unless in connection with aluminous compounds. He would call attention to the similarity in composition to the Bessemer slags formed in the air conversion of copper mattes, which also seem to consist mainly of singulo-silicates of iron, as shown by analyses which have been published.

It has been observed that when the silica in the charge got too high, the slag did not increase in acidity, but fragments of unmelted quartz began to escape from the tap-hole. Possibly a higher temperature at the tuyeres would have caused this excess of quartz to combine with the already formed silicates, giving rise to bi-silicates or more acid ones.

In many instances, as for example at Leadville and at Deadwood, limestone is added as a flux, the cost at the latter place being about 1*s.* per ton of ore smelted. But here the quarry of pure limestone is situated alongside the smelter, and the flux can be shot into trucks for hauling to the feeding floor. At Leadville, using 18 % of limestone, the cost is 1*s.* 5*d.* per ton of ore smelted.

It is contended by Lang, however, that the slags produced in pyritic smelting are as clean as any, even when the iron forms almost exclusively the base. Lime, magnesia, and baryta are regarded with equal favour by D. Doerffel.*

The average assay of the slags at Deadwood and at

* Jl. Chem. & Met. Soc. S. Africa, i. 49 (1898).

Mineral are stated at less than 1 dwt. gold, when the matte is raised to a maximum value of 50 oz. fine gold per ton.

An ingenious way of effecting the removal of slag automatically is to run the waste water from the furnace jackets against it, whereby it is granulated, and easily transported by the same stream of water to any convenient dumping ground, by means of a launder. Slag pots would need to be provided in case of accident.

*Fuel.*—In extreme cases, matte smelting can be conducted without any fuel whatever. One atom of the sulphur in the pyrites ($FeS_2$) being burned, the resulting FeS is decomposed in the presence of silica ($SiO_2$) and oxygen into oxide of iron and sulphurous acid (FeO and $SO_2$), the FeO combining at once with the $SiO_2$. These reactions are accompanied by an evolution of heat which is estimated to exceed 3000° F., and it is claimed that under the best conditions as high as 85% of the total available thermal units are utilised.

But generally some fuel is added. This for the most part takes the form of coke, but coal, sawn wood blocks, and gas may all be used. Charcoal is of small service, as it crushes to powder under the weight of the charge. In common practice, 1 to 8% of coke is added. At Keswick, Lang employed $2\frac{1}{2}$% coke, the cost amounting to 1*s.* $5\frac{1}{2}$*d.* per ton of ore; this served also for heating the blast. This authority considers that charges containing no more than one-third their weight of sulphides can be smelted with as little as $2\frac{1}{2}$% of coke, and yield a very high concentration, and under any ordinary circumstances 2 to 3*s.* per ton of charge ought to be the maximum outlay for fuel. At the Bi-metallic Co's. works, Leadville, the coke burned is $6\frac{1}{4}$% on the weight of the ore, and its use is mainly to support the finely-crushed concentrates; its cost, at 30*s.* per ton, is about 2*s.* per ton of ore smelted. At Mineral, Idaho, Lang used 7% of coke. At Mount Lyell, the consumption of coke is 6% on ore alone, or $2\frac{3}{4}$% on total charge.

*Furnace.*—The blast furnace may be of any approved type employed for matte smelting. As pyrites decrepitate in heat, and will pack so tightly as to choke a high furnace, a column of 8 to 10 ft. is quite sufficient. The coarser the ore the better,

and in no case can more than half the charge consist of fine concentrates. The furnace should be water-jacketed from top to bottom, and be fitted with matte separators at both ends.

At the Bi-metallic Co.'s works, an old lead furnace 80 × 36 in., altered and adapted, and provided with a hot blast at 750° F., was long used; but in 1898, it was replaced by a new matte furnace measuring 215 × 36 in. at the tuyeres, which is probably the largest of the kind in existence.

An ordinary blast furnace, such as is used for lead smelting, is not adapted for matte smelting, because of the deep crucible, and the comparatively high temperature at which matte solidifies; the matte chills as soon as it sinks below the smelting zone, and "freezes" so as to impede tapping. Hence in Hungary recourse has been had to the *spur-ofen*, which has no crucible, but the bottom made sloping to a tap-hole immediately below the tuyeres; this remains constantly open, the molten matte flowing into outside wells, where it sinks below the slag and is tapped into moulds, while the slag overflows continuously into the usual wheeled pots.

Either a hot or a cold blast may be used. For slags which do not readily fuse or are high in sulphur, a hot blast is necessary; when sulphur is low and lead is present, a cold blast is better. It is essential to have means at hand for producing a blast more than equal to the normal demands. Weakness of blast is sure to limit the degree of concentration, and may even result in a frozen charge.

*Cost.*—Very few reliable statements are published concerning the cost of matte smelting.

Lang gives the exceptionally low figure of 6*s.* 9½*d.* per ton of ore at Keswick, California, in 1897; and says that 12*s.* 6*d*, per ton will cover the smelting and matte refining at Mineral, Idaho.

At Leadville, the cost, in 1892, was stated as follows:—

| | *s.* | *d.* |
|---|---|---|
| Coke at 31*s.* 3*d.* per ton | 1 | 11½ |
| Oil for hot blast at 4*s.* 7*d.* | 2 | 0½ |
| Limestone, 18 % at 7*s.* 11*d.* | 1 | 5 |
| Labour and sundries | 10 | 11 |
| Total | 16 | 4 |

The charges at Deadwood, S. Dakota, in 1893 were quoted to the author at 32*s.* to 40*s.* per ton on the weight of ore delivered (wet or dry), and about 20*s.* a ton seemed to be approximately the actual cost. But Rickard says * that the Homestake concentrates are "treated at a charge of half their assay value," which he puts at 5 to 8 dollars a ton, making the charge 10*s.* 5*d.* to 16*s.* 8*d.* per ton.

Costs at Mount Lyell are officially stated at 14*s.* 2¼*d.* per ton.

*Results.*—The Deadwood smelter pays for 98 % of the assay in gold and 90 % of the silver, which implies pretty close working. It will be understood that the resulting product is not bullion, but that the gold and silver, with base metals such as copper, lead, nickel and cobalt, are concentrated into a matte. This is disposed of to the lead or copper smelters, who treat it as an ore.

Official ½-yearly figures (1899–00) show gold losses at Mount Lyell to be 1·26, 4·24, and 1·53 %, the lowest being on 2¼-oz.

**Metal Smelting.**—The smelting of copper and of lead are important special industries which cannot be dealt with in this volume. No gold mill ever embraces either of them.

But copper and lead smelters are nearly always buyers of auriferous concentrates and rich ore, which they subject to an oxidising roast, and thus obtain in a pulverulent condition an excellent flux (oxide of iron). There is practically no loss of the precious metals in an oxidising roast; they remain, in a freed state, among the oxide of iron, and are totally absorbed by the copper or lead in the blast furnace.

For the smelters' purposes, dry concentrates, especially if they have been sweet-roasted (see p. 361), are preferable to matte. The subject for consideration by the mill or mine producing a concentrate or pyritic ore is the relative cost of transport of the raw article on the one hand, and the diminished cost of transport plus the cost of sweet-roasting or of matte-smelting (as the case may be) on the other. There are also the smelters' charges to be taken into account, as they are based on the tonnage dealt with and not on the bullion contents.

* 'Stamp Milling of Gold Ores,' 1897.

Smelters' rates fluctuate very widely, being governed to a great extent by questions of demand, supply, and competition. All agree in making a heavy deduction for each unit (1 %) of antimony, zinc, or other volatile metal; and none pays for the first 5 oz. of silver per ton.

A common practice with American smelters is to pay for 95 % of the gold assay contents, and to make a "returning charge" according to the richness of the ore—thus say 18 to 35*s.* per short ton for stuff assaying 10 to 12 dwt. gold per ton; 50*s.* per ton on ore carrying 2 oz. gold and 8 to 10 oz. silver; and so on.

Colonial smelting works follow similar lines, thus:—

| Gold Assay of Ore. | Returning Charge. *s.* | *d.* | Per cent. of Gold paid for at 4*l.* per oz. |
|---|---|---|---|
| 4 oz. and under . . . . | 50 | 0 | 92½ |
| 4 to 6 oz. . . . . . | 50 | 0 | 93½ |
| 6 to 8 oz. . . . . . | 47 | 6 | 94½ |
| 8 oz. and over . . . . | 45 | 0 | 95 |

Approximate English rates are as follows, the ton used being the long one (of 2240 lb.):—

| Gold Assay of Ore. oz. | Price paid per oz. £ | *s.* | *d.* | Gold Assay of Ore. oz. | Price paid per oz. £ | *s.* | *d.* |
|---|---|---|---|---|---|---|---|
| 5 to 10 | 3 | 15 | 0 | 60 | 4 | 0 | 0 |
| 10 to 15 | 3 | 17 | 6 | 80 | 4 | 0 | 9 |
| 15 to 20 | 3 | 18 | 0 | 100 | 4 | 1 | 0 |
| 30 | 3 | 18 | 9 | 150 | 4 | 2 | 3 |
| 40 | 3 | 19 | 3 | over 150 | 4 | 2 | 6 |
| 50 | 3 | 19 | 9 | | | | |

The Mint value of pure gold is about 4*l.* 4*s.* 11¾*d.*. or say 4*l.* 5*s.* per oz. in round figures, so that on say 20-oz. ore the smelter gets a difference of practically 7*l.* a ton, while his loss probably does not amount to 1% or 4 dwt., say 17*s.* On 100-oz. ore, his loss or deficiency in recovery will be less in proportion, while the difference in actual value and price paid for the gold amounts to no less than 20*l.* a ton. Custom milling is a highly remunerative business, but it comes a long way behind smelting.

# CHAPTER XIII.

## COMPLETE SYSTEMS, RESULTS, AND COSTS.

THE great range of variation manifested by gold-bearing ores is responsible for the diversity of method adopted for recovery of the precious metal. In selecting a method or combination of methods for milling the product of any mine, very many points have to be taken into consideration. The desideratum sought in all cases is the same—viz. a maximum extraction at a minimum cost, but it is almost an impossibility in any instance to get the two conditions simultaneously, hence one or other must be sacrificed somewhat. Not only is the assay value of the ore an important factor in choice of treatment, but the condition of the gold is of equal moment —whether it be fine or coarse, whether free or contained in pyritic matter. The presence of other metals of value is another controlling influence, as for instance much silver alloyed with the gold, or the existence of copper, lead, or antimony in association with it. The nature of the gangue cannot be overlooked : it may be highly silicious and insoluble (quartz) ; or it may be calcareous and soluble ; or again, talcose or clayey, and prone to slime. Further, there will be local conditions, more particularly having reference to water supply (volume and quality), fuel (for power and for roasting), labour (both skilled and unskilled), transport, climate, and so on. The potential output and duration of the property, and the possibility or otherwise of catering for adjacent properties, will have some effect also. Custom or public milling has not been developed nearly as much as it deserves to be, and there remain many fields where a good modern mill could earn a

splendid revenue and do a great service to the mine owners at the same time.

Complete milling installations are of many kinds, embracing, besides the mere crushing which is a necessary feature in all, the following operations :—

(*a*) Amalgamation only.
(*b*) Amalgamation and concentration.
(*c*) Concentration only.
(*d*) Amalgamation, concentration, and chlorination.
(*e*) Chlorination only.
(*f*) Amalgamation, chlorination, and cyanidation
(*g*) Amalgamation and cyanidation.
(*h*) Cyanidation only.

Finally some reference may be made to subsidiary operations designed for recovery of base metals contained in some ores.

**Amalgamation only.**—*Mills.*—An example of an amalgamation mill using stamps and plates, as built by Fraser & Chalmers, is shown in Fig. 190. The ore, brought in on an elevated road by truck *a*, is tipped on the grizzly *b*, so that the fines may at once pass to the battery bin *d*; the remainder falls on a floor, and is shovelled into the swing-jaw breaker *c*. This is so situated as to convey considerable vibratory movement to the battery framing which is united to the bin, and, coupled with the tendency of the continually falling ore to thrust the bin forward, is liable to result in a gradual outward lean of the battery, which is not desirable. With the very short centres for the driving belts, they all need tighteners, that for the breaker being controlled by handwheel *e* and lever *f*. The ore gravitates from the bin *d* to the feeder *g*, and so to the battery *h*. An amalgamating table is provided at *i*, followed by a mercury drop *k*. No windows are indicated, but they should be freely provided at *l* for illuminating the front of the battery. The breaker is so placed that daylight can only reach it apparently by a window in the roof at *m*. The mill is constructed of timber

framing and sides, with corrugated galvanised iron roofing. The main shafting lies on the ground sills at *n*.

A structure on similar lines by the Sandycroft Foundry Co., is shown in Fig. 191. A double track *a* leads to the grizzly *b*, which feeds the swing-jaw breaker *c*. As in the case of the previous example, no provision is made for

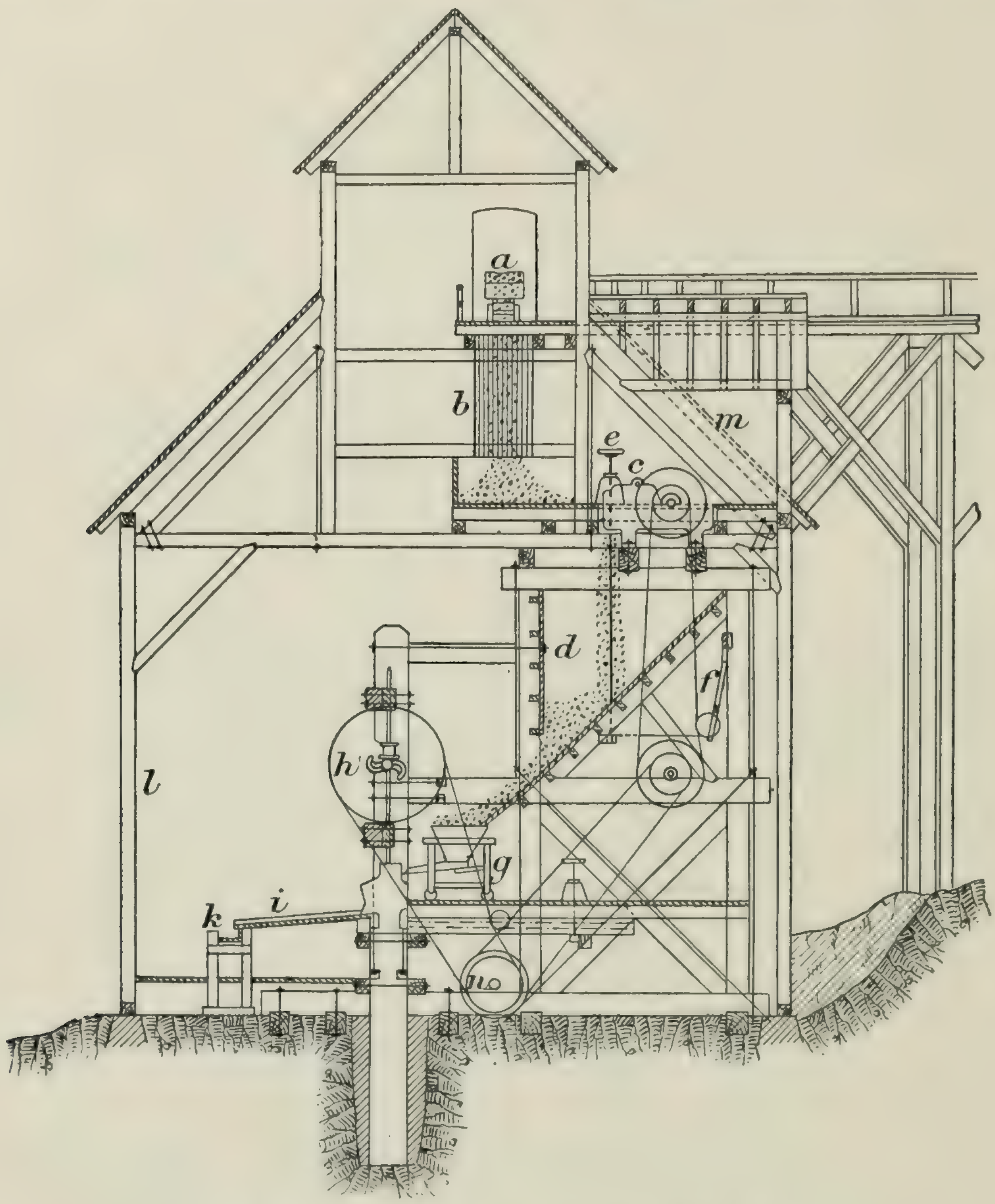



FIG. 190.—AN AMALGAMATION MILL.

accumulating unbroken ore,—it must be broken as fast as delivered, which often entails great inconvenience. The battery bin *d* discharges into feeder *e*, supplying battery *f*. Owing to short centres, the driving-belt of the last-named is provided with a tightener *g*. An overhead crawl *h* is supported from the joists *i*, which also carry rods *k* sustaining a narrow cam-floor *l*. Water is supplied through pipes *m*,

while *n* is an amalgamated copper-plate table. Stairways give access to all floors, and plenty of window area is furnished.

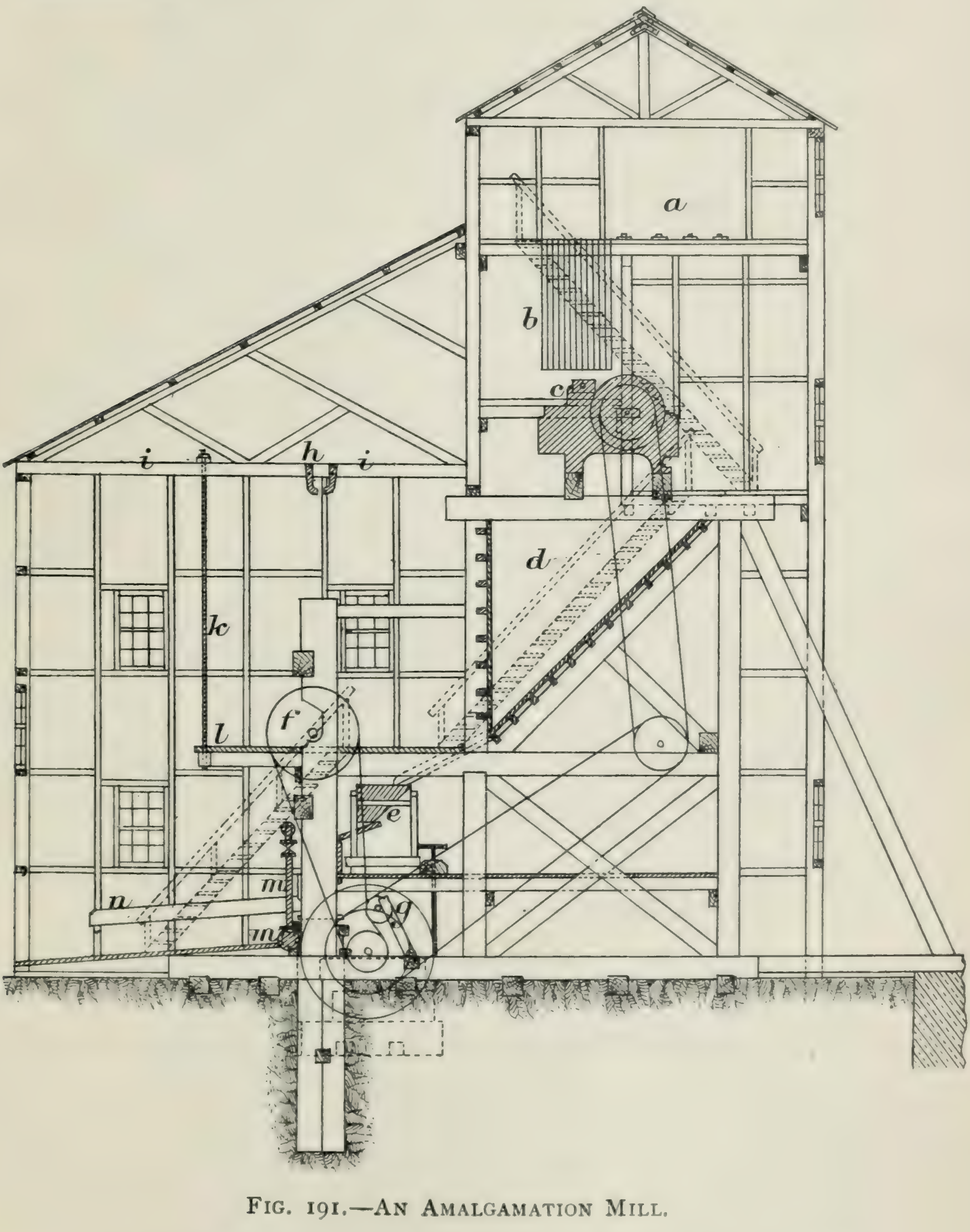



FIG. 191.—AN AMALGAMATION MILL.

In both these installations, a gyratory breaker would be much preferable, causing far less vibration, affording a finer

product, and demanding less labour for feeding—in fact it may be made almost self-feeding with a suitable shoot.

There are various advantages in arranging batteries back to back, when the number of stamps exceeds what can be

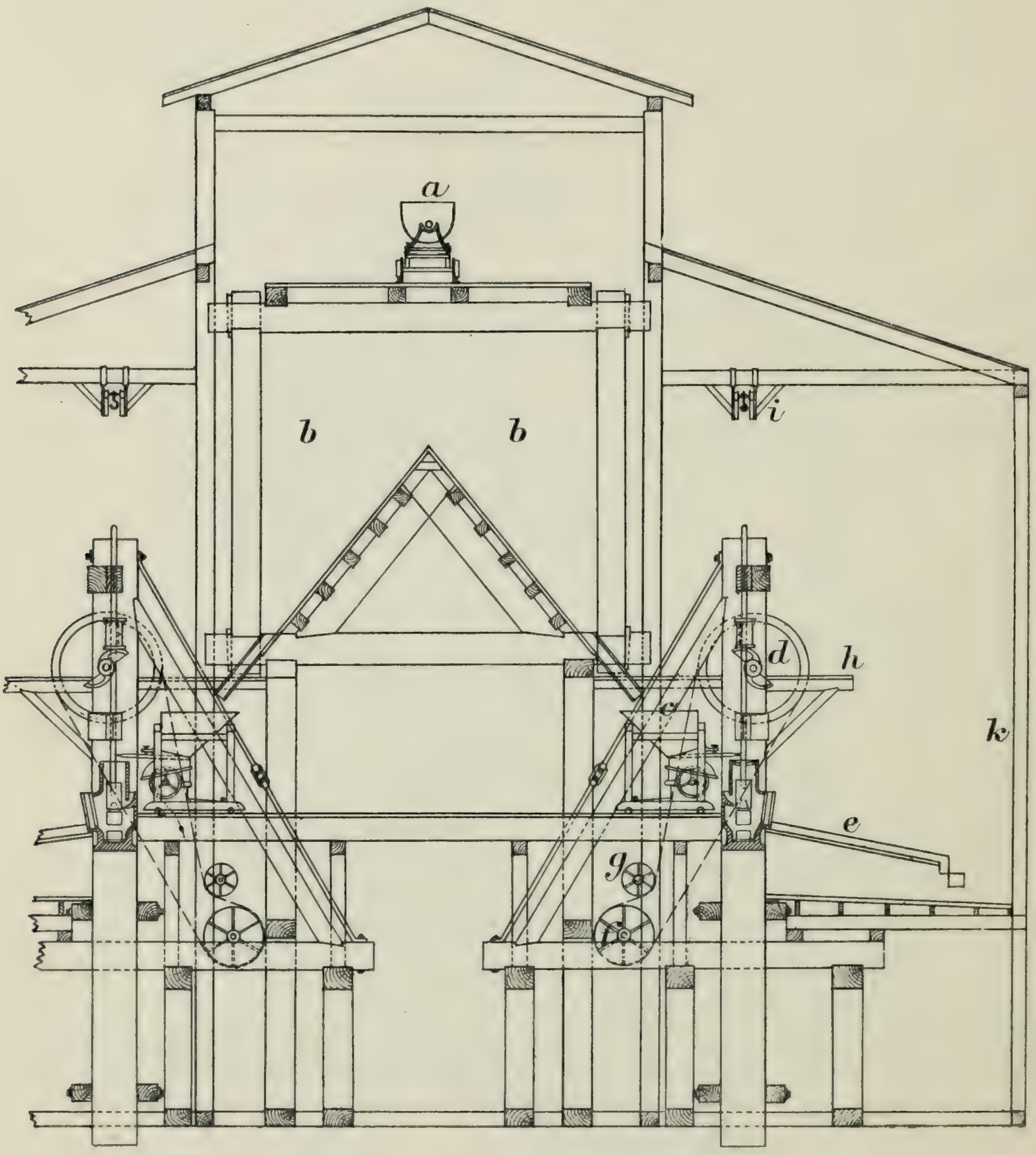



Fig. 192.—An Amalgamation Mill.

satisfactorily controlled by one man. This is generally reckoned at 60 as a maximum. An example of such an arrangement is afforded by the Haile mill (S. Carolina), built by the Mecklenburg Ironworks, and illustrated in Fig. 192. The side-tipping truck *a* furnishes ore to the double bin *b*,

being capable of discharging on either hand. As will be seen, there is a very material increase of bin accommodation. In this case, either the ore is delivered unbroken to the battery, a most reprehensible practice ; or it is broken by machines erected on a separate site and driven by another motor, which is distinctly the best arrangement when the scale of operations justifies it. The details of feeder *c*, battery *d*, amalgamating

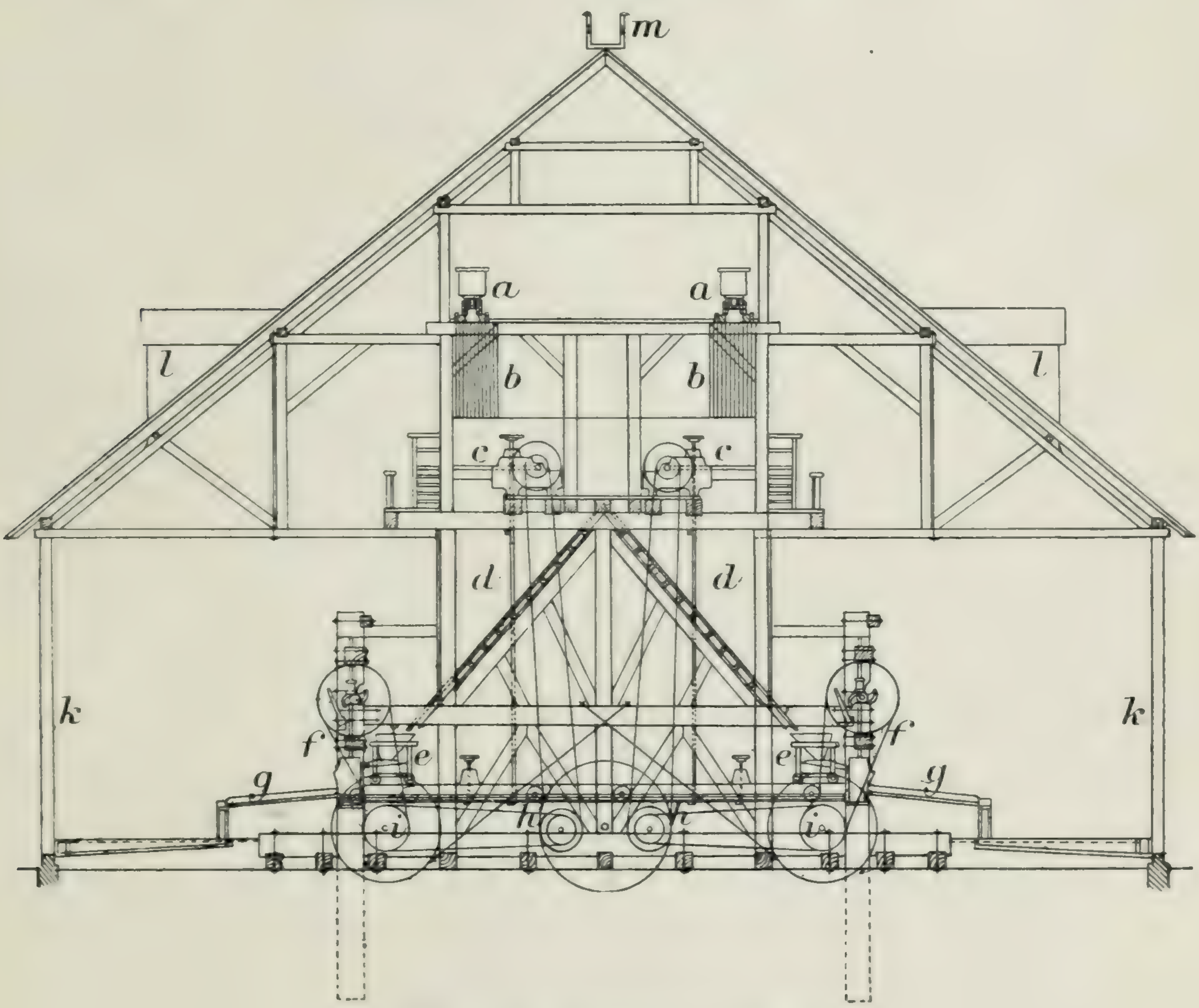



FIG. 193.—AN AMALGAMATION MILL.

table *e*, line-shafting *f*, tightener *g*, cam-floor *h*, and crawl *i* call for no special comment. But it will be noticed that the whole structure is carried at an abnormally high elevation above ground level, or that a lavish use of pile timber has been made in the foundation without any obvious purpose. No windows are shown at *k*, but they are indispensable.

Another example of the back-to-back system is the 120-stamp mill of the Highland Co., Deadwood, S. Dakota (Fig. 193) built by Fraser & Chalmers. A separate set of

swing-jaw breakers *c* is provided for each line of batteries, served by trucks *a* and grizzlies *b*, and delivering to bins *d*. The feeders *e*, batteries *f* and amalgamating tables *g*, with belt tighteners, are virtually the same as in their single system (Fig. 190); but the line-shafting *h* is thrown more to the rear, so that the breaker is driven direct while the battery needs a counter-shaft *i*. Light is admitted to the batteries by windows in the wall *k*, and to the breakers at *l*. On the crown of the roof is a water-trough *m*, very useful in case of fire. No bin is provided for the breakers and a very small one for the stamps, considering their high duty (about 4 t. per head per 24 hours), and in this respect the arrangement compares unfavourably with Fig. 192. Gyratory breakers would be much preferable to swing-jaws, especially in such a position.

The typical Transvaal mill, shown in Fig. 194, differs materially in its bin arrangements. Breaking is performed outside the mill proper, and the broken ore is brought in by a double overhead track *a*, carrying trucks which empty into parallel bins *b* with A frames *c* at intervals to support the tracks. These bins have square sides and a flat floor, which greatly increases their holding capacity; and though the reserve of ore represented by the dotted line *d* cannot be availed of without manual labour to pass it to the feeder *e*, it is nevertheless accessible, and it always serves a useful purpose in protecting the floor from the constant feed, and thus increasing its durability. There is abundant provision of light in this mill by windows *f*, and of ventilation by revolving windows at *g*. The design of the mill is due to G. B. Connor, and it is adopted at the Simmer and Jack, and other large undertakings.

*Methods and Results.*—A Chilian example of amalgamation milling is described by G. M. Barber.* In the district of Guanaco the gold generally exists in a microscopic state of division, disseminated throughout hard quartz, and exceedingly fine grinding is necessary to liberate it; for this reason, and also on account of the extreme hardness of the quartz enclosing it, the mineral presents great difficulties to suc-

* Trans. Inst. M. & M., v. 99 (1897).

cessful grinding and amalgamation. The gold is free. In some cases, silver also occurs in the ore, but in a mineralised condition. As there is no fresh-water supply, sea-water is

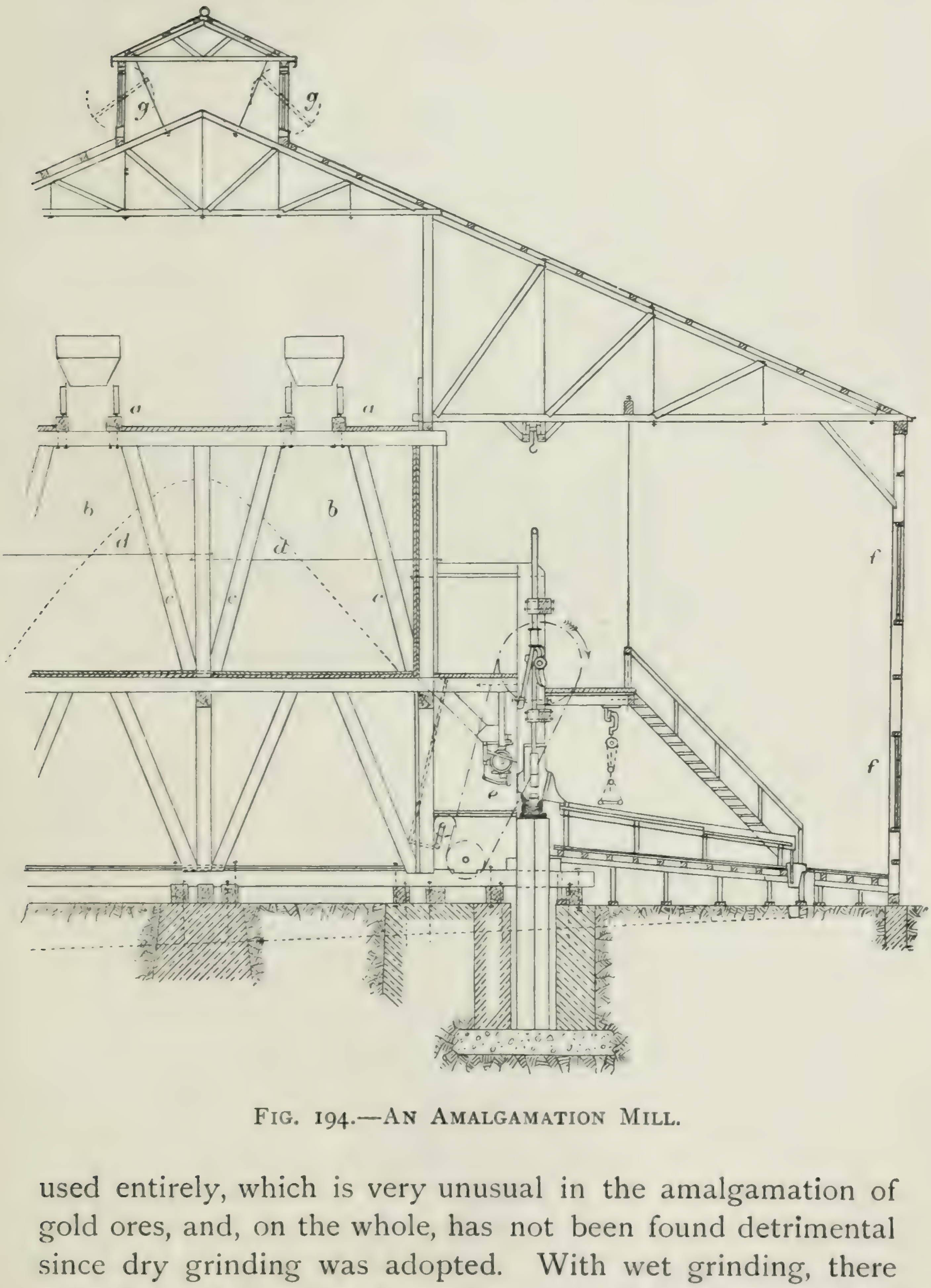


FIG. 194.—AN AMALGAMATION MILL.

used entirely, which is very unusual in the amalgamation of gold ores, and, on the whole, has not been found detrimental since dry grinding was adopted. With wet grinding, there was a great loss of gold, most probably in the form of float

gold, due to the exceptional density of the water. The adoption of dry grinding (in ball mills) and subsequent pan amalgamation of the dry-ground material (p. 257) has most successfully overcome the difficulty. The case is a very interesting one, but no statistics of cost are given. The ore contains nearly 37 dwt. gold and about 5 oz. silver per ton, and the gold extraction attained is under 76 %, so that 9 dwt. or some 36*s.* worth of gold per ton is lost in the tailings. This must be accounted very poor work, and it seems an open question whether cyanidation on the one hand and smelting on the other would not accomplish much better work at but little greater cost.

The mills of all the Indian companies have for years depended upon amalgamation, and until quite recently upon it alone, battery treatment being followed by pan-grinding. But it is difficult to arrive at an exact estimate of the merits of their work from the statistics published. The gold appears to be of high standard, the fineness of their bullion being given at 920, and for this reason its arrest is somewhat facilitated. It is probably also both coarse and free, the battery saving being high, and no concentrating machinery being used for collection of sulphurets.

The Mysore Co., in 1896, recovered 25¾ dwt. bullion per ton from battery treatment, at a milling cost of just over 5*s.* per ton, which, on a 90-stamp establishment and with cheap native labour, seems a very high figure. A further recovery of 4½ dwt. per ton was made from all tailings subjected to pan grinding, at an additional cost of 6*s.* 10*d.* per ton treated. In 1898, the extractions were 29·58 dwt. and 4·88 dwt. respectively

The Nundydroog Co., with 40 head, in 1896 recovered 19½ dwt. bullion per ton in the battery at a cost of 4*s.* 4*d.* per ton; and 3½ dwt. from each ton of tailings subjected to pan amalgamation, at a cost of 6*s.* 3*d.* per ton treated.

The Ooregum Co., in 1898, got an extraction of 11¾ dwt. in the battery and 2½ dwt. in the pans.

Since cyanidation has proved successful, the pan grinding of tailings in Indian mills is going out of use.

The Mexican mine of Mesquital affords an ore well adapted for amalgamation, being free from sulphurets of any kind, and assaying 10 dwt. gold and 16 dwt. silver per ton. But while the ore is essentially quartzose and non-pyritic, the gangue contains much ferruginous clay, making a slime that necessitates an unusual volume of water in the battery, and doubtless involves some loss of the fine gold, especially as its specific gravity is lessened by the high alloy of silver. Thus the average assay of the tailings is 2 to 2½ dwt. gold per ton, equivalent to a loss of 20 to 25 %. Inasmuch as 72 % of the tailings will pass a 100-mesh screen it would almost seem as if a mistake is being made in the system of treatment. If very much coarser crushing were adopted, aiming at catching only the heavy gold in the battery, the duty of the stamps could probably be doubled, and the cost of stamping halved; the resultant pulp would then in all likelihood be quite amenable to cyanidation. In that way it might be possible to extract an extra 1 or 1½ dwt. without additional cost. The present cost, running 50 stamps with a duty of 2·2 t., and enjoying very cheap labour, is 4*s.* 10*d.* a ton, which is not a low figure by any means.

Among United States mills there is an interesting case at the Idaho mine, Alabama, where a Huntington mill is used in conjunction with a stamp battery.* The ore is a partially decomposed hornblende schist carrying free gold, much of which is in fine flaky colours, difficult to save because of the great volume of water needed in the battery. To remedy this, the ore is passed over a grizzly, which in a great measure separates the hard undecomposed rock from the clayey decomposed portion; the former then passes through a breaker to the battery, while the latter goes to the Huntington mill. The arrangement works very well. Shaking copper-plate tables, riffled stationary plates, and blanket strakes are all employed as supplementary amalgam catchers, but no figures are submitted as to relative duties or costs.

The 60-stamp mill at El Callao, Venezuela, operating on a hard quartz, in 1899 had a duty of about 2·5 t. per head,

* J. Franklin, En. & Min. Jl., May 15, 1897.

and crushed to 40-mesh at a cost of 6*s*. 6*d*. per ton, the fuel consumption being equivalent to about 22 lb. of good coal per ton milled. The loss in tailings is given at 2¾ dwt. per ton.

Simple battery treatment at the Hannan's Reward mill, W. Australia, in 1899, cost about 15*s*. 2*d*. a ton.

While amalgamation remains in occupation of the most prominent position among gold-saving methods, there is a general recognition of the fact that its true scope is for saving coarse free gold only, and that in other spheres it is surpassed by more modern processes. Consequently there are very few mills in which amalgamation is alone relied on, and fewer still where such reliance is justified. As a ready means of collecting heavy gold particles, it is unrivalled in simplicity and efficiency; but virtually every ore carries some value in very fine free gold, or in pyritic matter, or in both, and therefore some supplementary treatment is called for.

**Amalgamation and Concentration.** *Mills.*—In Fig. 195 is illustrated an amalgamating and concentrating mill, by the Mecklenburg Iron Works, N. Carolina, which is evidently designed for a climate where heavy snow-falls are unknown, judging by the flat pitch of the roof. It comprises a grizzly *a*, swing-jaw breaker *b*, battery *c*, plates *d*, classifiers *e*, and two rows of vanners *f*. There is no bin for the breaker. The latter is well placed for solidity of foundation; but the distance between centres of countershaft-pulley *g* and breaker-pulley is so short that without a tightener the belt would be continually slipping. The battery-frame is built independent of the bin timbers, which is a good point in avoiding thrust and vibration. The battery driving-belt again is very short, and it would seem as if the line-shafting *h* might have been better placed at *i*, or even farther to the rear, when the breaker counter-shaft might have been dispensed with. The vanners are under the disadvantage of taking their motive power from the same source as the breaker and battery; this means that they are subjected to constant variation of speed which is fatal to good work.

In Fig. 196, showing a mill by Fraser & Chalmers, grizzly *a* feeds a gyratory breaker *b*, and the battery-bin *c*

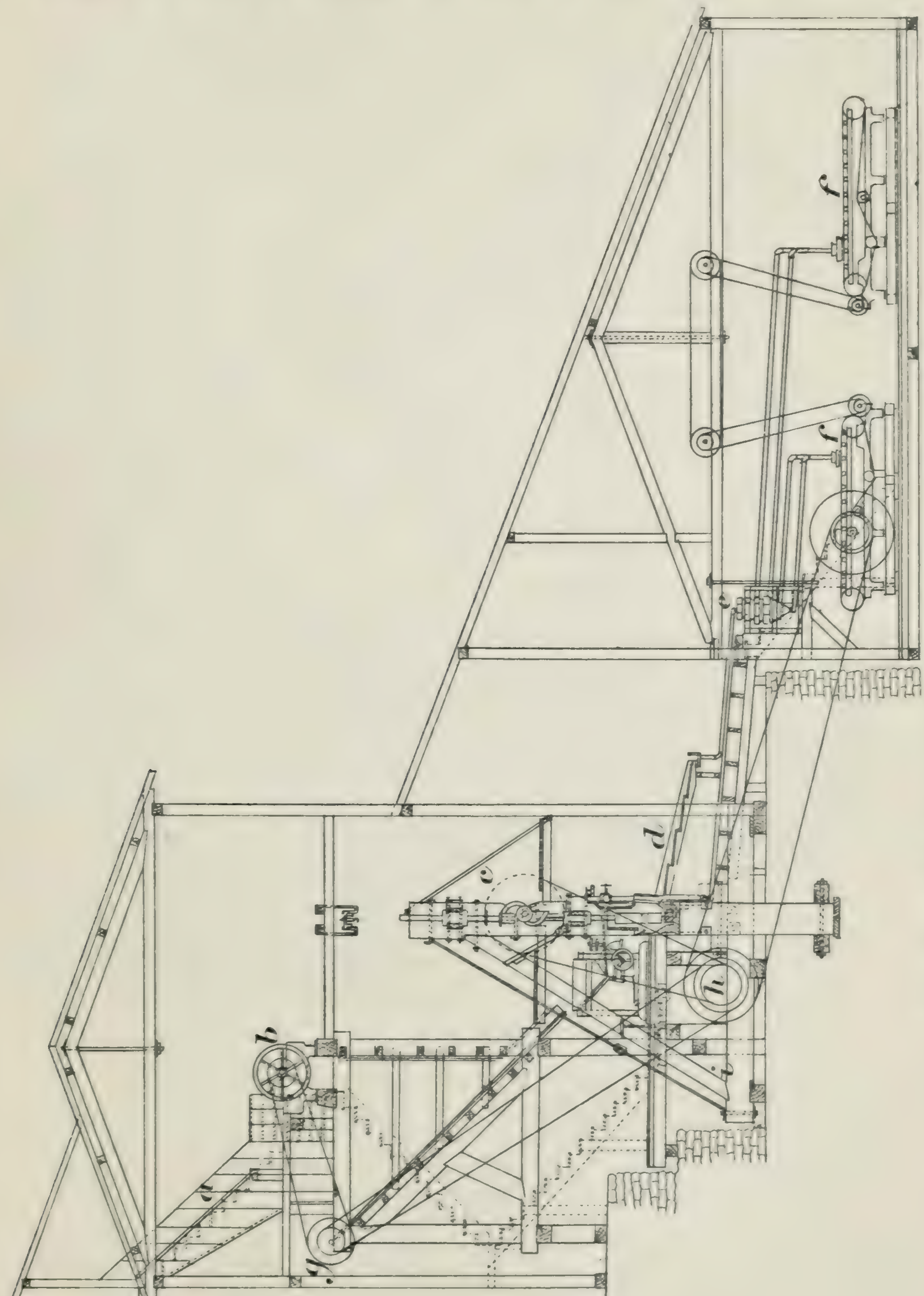



FIG. 195.—AN AMALGAMATION AND CONCENTRATION MILL.

is much more capacious. There is the usual short centering of belts, and common source of power for all operations.

A classifier *d* properly intervenes between plates *e* and vanners *f*.

A mill by the E. P. Allis Co. is represented in Fig. 197: *a*, grizzly; *b*, swing-jaw breaker; *c*, bin; *d*, battery; *e*, plates; *f*, classifiers; *g*, vanners in a double row. The roof *h* over

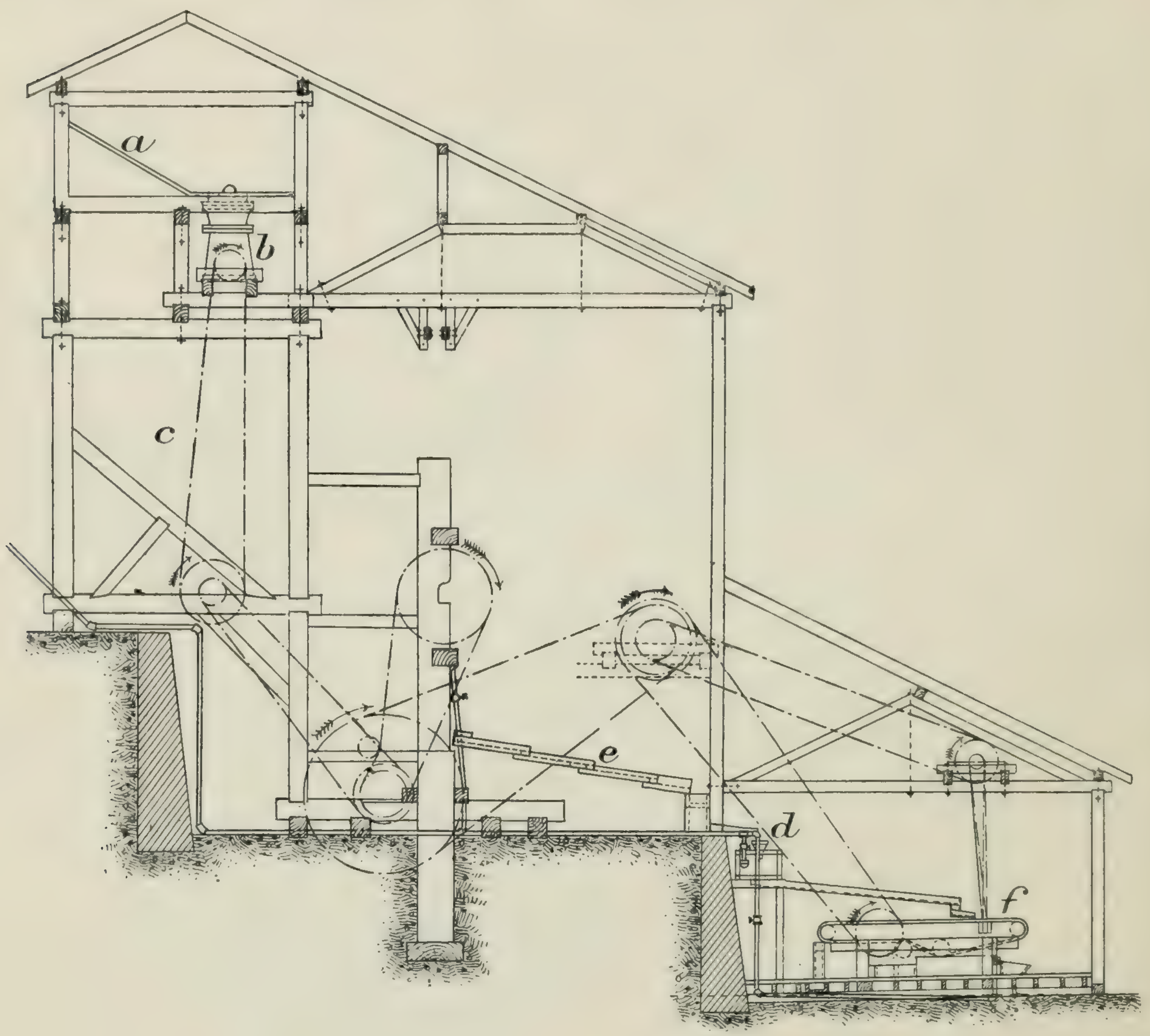



FIG. 196.—AN AMALGAMATION AND CONCENTRATION MILL.

truckway is quite unnecessarily elevated, wasting material and involving excessive exposure to high winds.

The Montana Co.'s 60-stamp mill, built by Fraser & Chalmers, and shown in Fig. 198, possesses the essential difference that it provides abundant bin space *a* for unbroken ore. The delivery from this bin, which is a feature that all mills ought to possess, is virtually automatic to the grizzly *b* and swing-jaw breaker *c*; but the battery-bin *d* is unusually

small. The delivery from plates *e* to the two rows of vanners *f* is direct, no classifiers being provided. A single motor

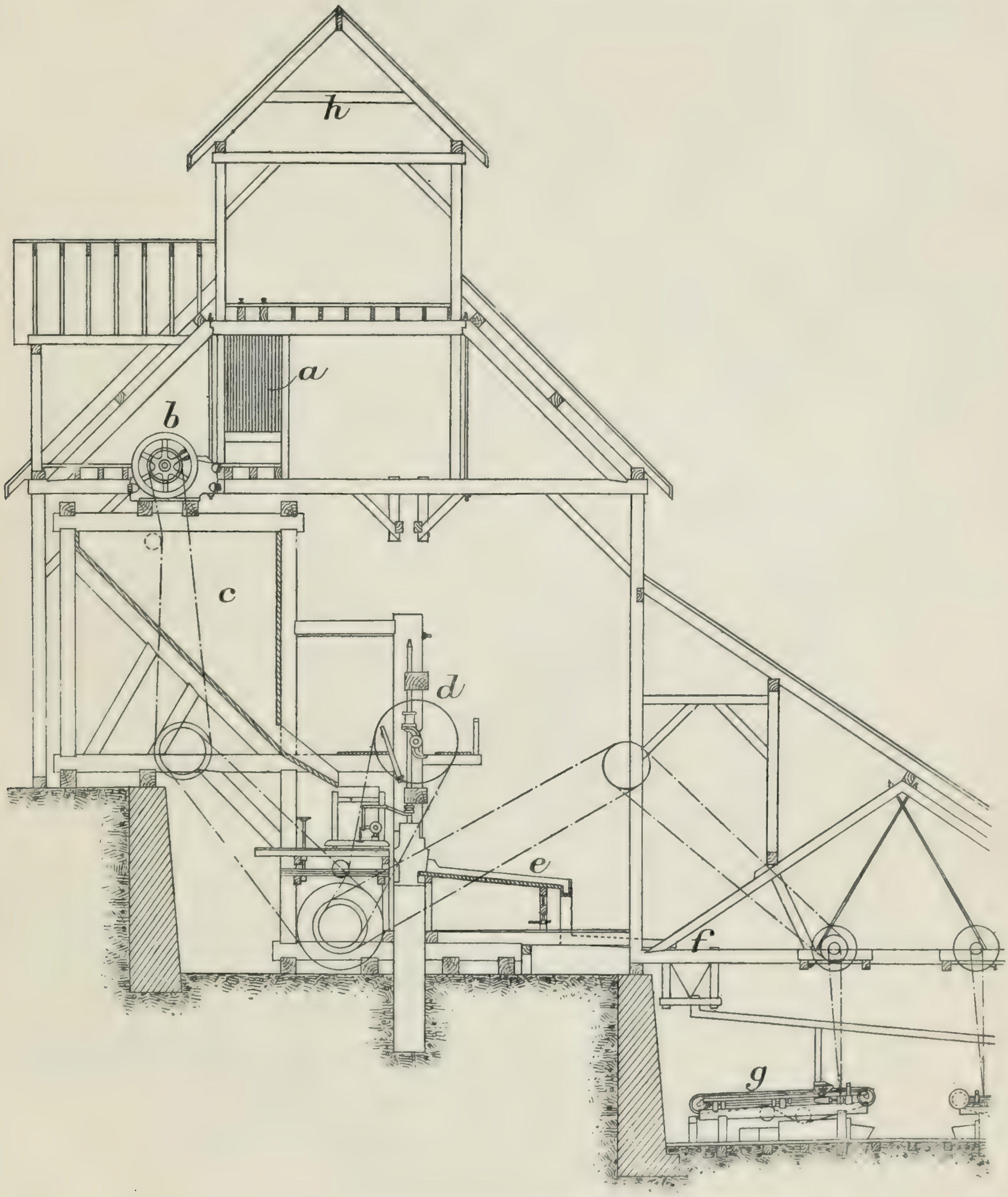



FIG. 197.—AN AMALGAMATION AND CONCENTRATION MILL.

drives everything, and the shortness of belts is again most prominent. The line-shafting *g* would be much better at *h*. The thrust of bin *d* and vibration caused by breaker *c* will be accentuated by the latter's position.

The 60-stamp (making a total of 120) mill of the Alaska Mexican Co., by the same firm, is built on a rock foundation against the embankment. The mill building is 88 × 114 ft., the roof sloping upward from the vanner-room to above the

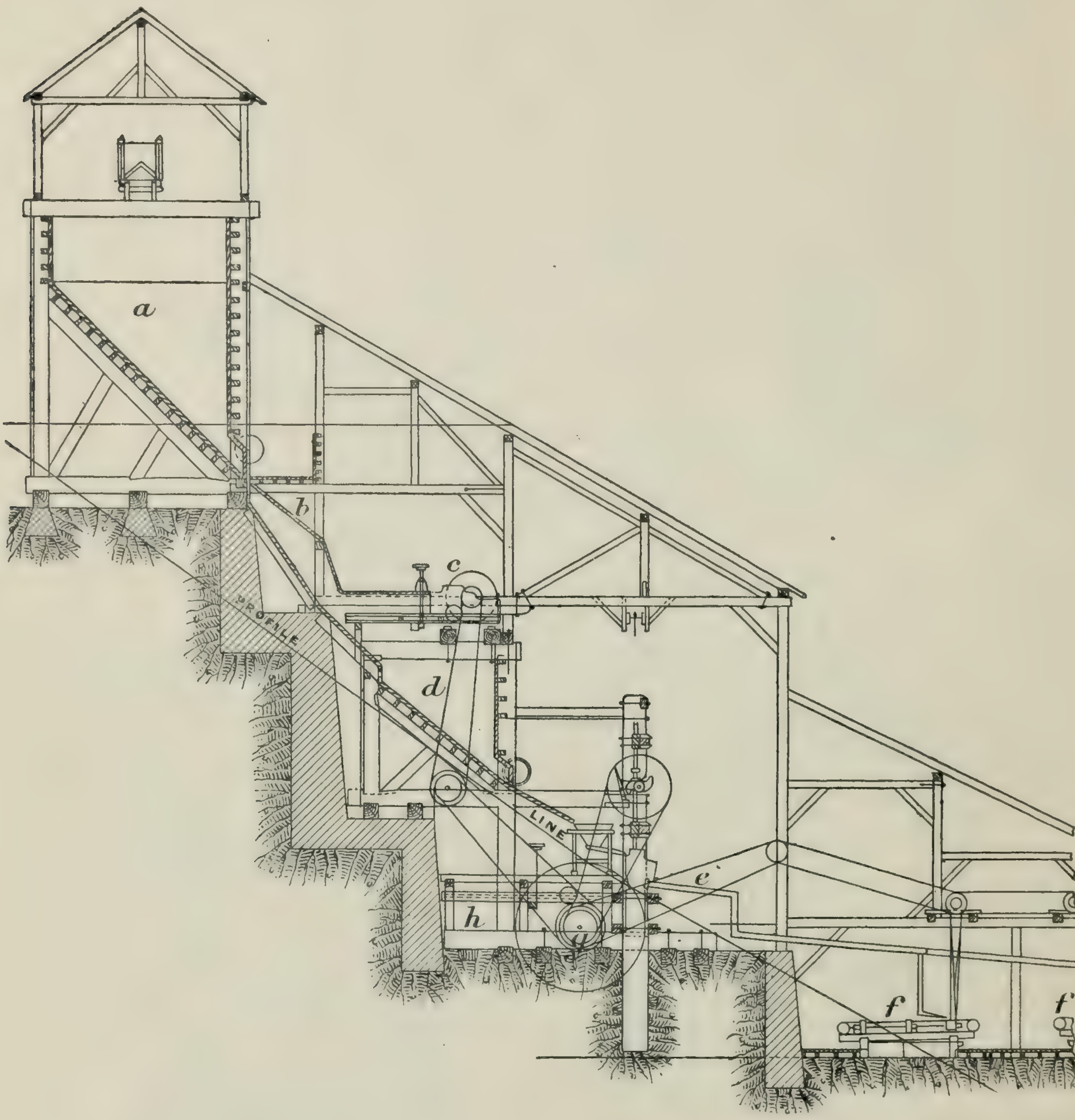



FIG. 198.—AN AMALGAMATION AND CONCENTRATION MILL.

ore-bin and battery, a distance of 63 ft. in height. The mill is double-boarded, with tar-paper lining between the boards and under the roofing. The outside is coated with red mineral paint. The boiler room, 36 × 40 ft., contains 3 tubular boilers, each 54 in. diam. and 16 ft. long, and having in all a capacity of 240 h.p. The coal bunker, located in front of the boiler, is

40 × 40 ft., and 21 ft. high, capable of holding 800 t. of coal, which is dumped from cars run in from the outside upon an inclined track. The engine room, closely sealed in by tongued and grooved timber, to keep out dust, is 36 × 40 ft.; height of ceiling, 18 ft. The driving engine is a twin compound condensing Corliss, the cylinders being 16 and 26 in. diam. respectively by 42 in. stroke. One engine cylinder is high- and the other low-pressure. The iron band wheel is 18 ft. diam. weighing 14 t. Steam power is only used in the middle of winter, when the supply of water is insufficient to run the mill. The engine develops 250 h.p. A Pelton wheel, 71 in diam., with water pressure at 180 lb. per sq. in., is amply sufficient to run the mill under full headway, and has power to spare to run an increased number of stamps.

The battery room is 26 × 114 ft., 35 ft. high, being 10 ft. above the vanner room. The battery is placed on the solid rock of the embankment, so that there is no possibility of any shaking or settling down of the foundations. The framework is built of Washington fir, and bolted in a framing to the ore bin by iron bars. There are 3 battery posts to a section (10 head), each 12 × 24 in., and two guide girths each 12 × 16 in. The knee beams fastening the battery frame to the ore bin are each 12 × 16 in.

An immense ore-bin, 14 × 114 ft. in plan and 20 ft. deep, is above and behind the battery, resting on a massive framework of timbers, built from a rock foundation. The bottom is constructed on an inclined plane, so that the ore slips through the shoots into the feeders by gravitation. A covered track-way for the ore-cars leads from the independent crusher house to the top of the ore-bin over its entire length. In the vanner room, 42 × 114 ft. in plan and 12 ft. high, are 24 Frue concentrators with plain rubber belts. The tables have each a small amalgam sill, an improvement lately added. The mill is lighted throughout with 90 incandescent electric lights of 16 c.p. each.

The crusher or breaker house is separate from the mill, and is 110 ft. high from the ground. Two gyratory breakers (with independent motor) each capable of crushing sufficient

ore for the battery, are securely fastened upon a strong and heavily-timbered "square" frame-work 27 × 31 ft. An automatic Reedy elevator hoists the cars to an ore-shoot which conveys the ore to the crushers. The ore drops from the crushers into the cars which carry it to the mill ore-bin.

Where the crushing and amalgamating machinery employed consists of Huntington mills, Fraser & Chalmers adopt the plan shown in Fig. 199. The grizzly *a* delivers to swing-jaw breaker *b* and bin *c*, feeder *d* passing the broken

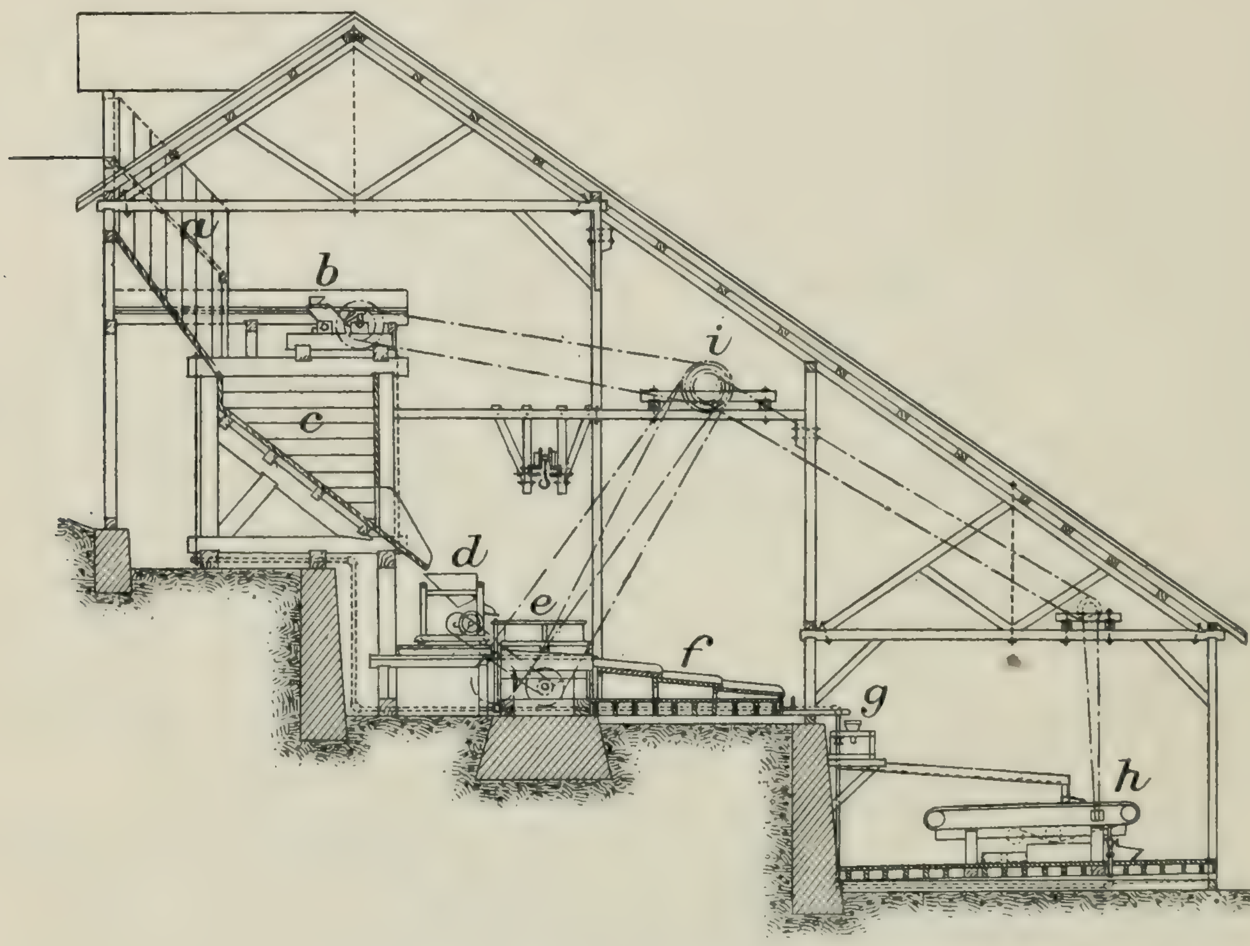



FIG. 199.—HUNTINGTON MILL PLANT.

ore on to the Huntington mill *e*. The pulp flows over amalgamated plates *f*, through classifiers *g* to vanners *h*. The situation of the line-shafting *i*, which drives the whole of the machinery, is exceptionally elevated.

An installation embracing steam stamps instead of the ordinary gravitation battery, designed by the Gates Iron Works, is illustrated in Fig. 200. The grizzly *a* delivers right into the hopper of a gyratory breaker *b*, but why the latter is perched upon a staging is not obvious ; if all that height is necessary, the bin *c* might have the advantage of it.

This last feeds the steam stamps *d*, which have outside plates *e*, whence a launder conveys the pulp to vanners *f*. From the illustration, it would appear that separate motors are provided for the breaker and the vanners.

In the whole of the examples presented there are indica-

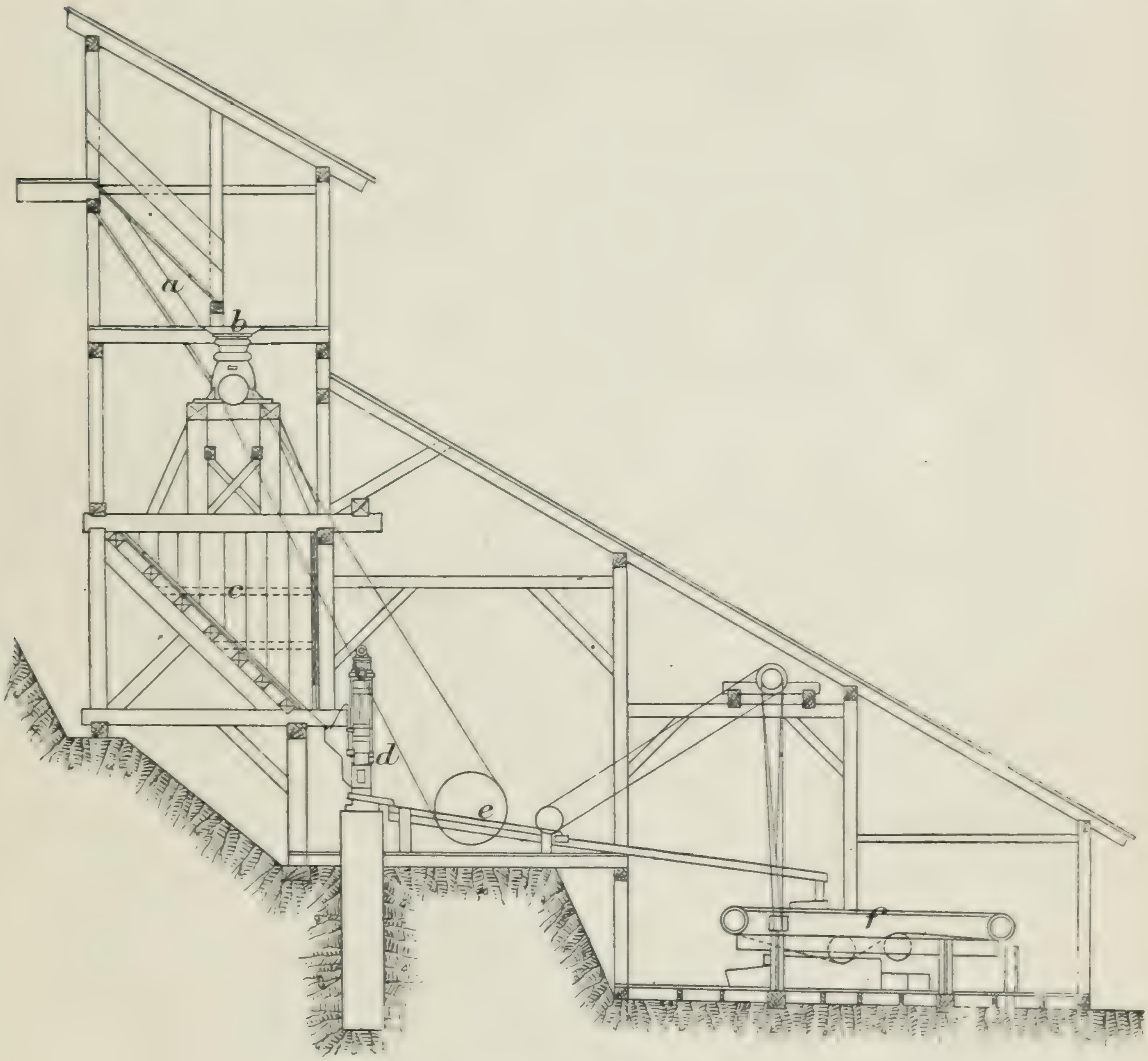



FIG. 200.—STEAM STAMP PLANT.

tions that board floors are used in battery and vanner room. This is a most unsuitable arrangement. Brick, or better concrete with good cement face, is eminently superior for both.

*Methods and Results.*—For many years the treatment of the refractory low-grade ores of some of the Brazilian mines was confined to very imperfect amalgamation and a rudimentary concentration of mineral for chlorination. These

ores consist largely of arsenical iron pyrites (mispickel) and their treatment is rendered more difficult by the presence of sulphide of bismuth. At one time the more pyritic portion would assay as high as 32 oz. per ton, but the average milling dirt does not now exceed 3½ to 6½ dwt. Stamp milling to 40 mesh, blanket concentration, and barrel amalgamation never succeeded in extracting more than about 60 % of the total value contained in the ore, and 80 to 85 % was the best that could be got from the rich concentrates. Obviously, such an ore was never adapted to mercurial treatment at all, and it is not surprising to learn that it has been eliminated, and the great cost for mercury, sickened by the arseniates and sulphates, is no longer incurred. The present method is to crush for concentration and to chlorinate the concentrates.

The auriferous ores of Remedios, Colombia, contain 1 to 12 % of sulphides, embracing arsenical and antimonial pyrites, galena and blende, while copper pyrites, molybdate of lead and native bismuth are occasionally met with, and more rarely tellurium. At the Frontino and Bolivia Co.'s mines,* from surface down to 220 ft. it was comparatively free milling, so that in years gone by it was possible to recover 60 % of its gold contents by a crude system of plate amalgamation in mills of the native type. The mineral subsequently became more refractory, so that amalgamation, except of concentrated pyrites in arrastras, had to be abandoned, and blanket concentration, the usual practice of the district, was substituted. In 1894, however, from 400 ft. downwards the ore was found to have become more amenable to amalgamation, which process has accordingly been re-instated in a mill of the Californian pattern.

The ore as it comes from the mine passes through grizzlies to a 10 × 4-in. Blake-Marsden crusher driven by countershafting. It falls into bins, of which there is one for every two batteries of 4 stamps, each capable of holding 40 t. A reserve bin with a capacity of 200 t. outside the mill, is constantly kept full, in order to avoid stoppage, should a temporary breakdown occur in the mine. Challenge feeders are

* F. Owen, Trans. Inst. M. & M., iv. 8 (1896).

used. There are no plates inside the battery. The four outside plates are each 4 ft. long by 2 ft. 6 in. wide, with an inclination of 1 in 20, and are silvered. From these the pulp passes over Frue vanners, one to each battery of 4 stamps, driven by countershafting from an auxiliary water-wheel at 195 rev. per min. After leaving the vanners, the pulp runs over blanket strakes, 12 ft. long by 2 ft. wide, with four divisions, and an inclination of 1 in. per foot; and thence into convex settling-pits 5 ft. diam. by 1 ft. 6 in. deep, after which it is led into wooden launders 50 ft. long, 2 ft. 6 in. wide, and 2 ft. deep. One launder is allowed to fill, whilst the other is being cleaned out. The tailings from these launders are allowed to run away down a stream. Here some natives have two small arrastras, driven by a hurdy-gurdy wheel, in which they amalgamate such of the pyrites out of these tailings as they can catch in the stream, paying a royalty of 25 % to the Company. The settling-pits are cleaned out daily, and the contents are washed on inclined wooden tables to extract a certain amount of mineral. It is in contemplation to roast them for barrel amalgamation, instead of shipping to smelters. The tailings caught in the launders outside the mill are dumped for future treatment, probably by cyanide.

The gold extraction by blanket concentration in 1892 was 60 % or $16\frac{1}{2}$ out of $27\frac{1}{4}$ dwt.

The gold extraction by plate amalgamation in 1894 was $38\frac{1}{3}$ % or $7\frac{3}{4}$ out of $11\frac{3}{4}$ dwt.

The fineness of the bullion contained in the milling dirt is about 628 gold and 372 silver, while the bar bullion as shipped is about 550 and 350.

An Australian example * of low milling-cost is afforded by the Myalls United mill, New South Wales. The plant consists of a 40-stamp mill (850 lb.), with amalgamating tables, blanket strakes, two Berdan pans, and a White-Denny pan for grinding concentrates. Steam power is used, and wood fuel costing 7*s.* a cord. Wages are 7*s.* 6*d.* to 9*s.* per 8-hour shift. From a Blake breaker, the ore is taken by Challenge feeders and is milled through punched screens

* W. R Thomas, Trans. Inst. M. & M., vii. 145 (1898).

with 200 holes per sq. in., the stamp duty being 2·53 to 2·81 t. After passing the plates and blankets, the tailings are elevated 30 ft. by two 12-in. plunger pumps driven from the cam-shaft by rope gearing. From 5·35 dwt., 3·57 dwt. is recovered in the battery for a cost ranging from 1*s.* 5·65*d.* to 1*s.* 10·41*d.* The average for 6 months is 1*s.* 7·35*d.*, made up by: labour and supervision, 11·38*d.*; supplies and repairs, 4·36*d.*; and fuel, 3·61*d.* per ton.

At the Wentworth and Aladdin's Lamp mines, Lucknow, N.S.W., the breakers are separately driven and placed on bed-rock, with bins holding 2 days' supply; they ordinarily run only 8 hours out of the 24. The stone after passing through them is raised by a hoist to the top of the battery building and tipped into the bins (which accommodate 4 days' consumption) behind the stamps. The dirt is thence fed into the mortar-boxes by Templeton rollers. Inside amalgamation is gained by means of housing-plates, and by dropping small measures of mercury into the mortars at intervals during the shift. 850-lb. stamps are used, a weight that is hardly sufficient to cope with the exceeding toughness of some of the rock; but a greater weight would slime the less hard and generally richer stone, and give an increased percentage of fine pulp in the tailings. The pulp travels over usual copper-plate tables, 16 × 4½ ft., the newer ones having two 3-in. besides the lip-plate drop. From the mercury trap at the foot of these, the pulp passes direct to Frue vanners, two being provided for each 5-head battery. The value of the pulp as it passes on to the vanners ranges from ½ to 3 oz. bullion, and the concentrates vary accordingly. Where much pyrrhotite or mundic is carried, these latter may only yield 6 to 8 oz. bullion per ton; when free from these minerals, the arsenical pyrites caught on the vanners run up to as high a value as 60 and even 100 oz. bullion per ton—an example having been recorded giving 172 oz. These are dried for shipment to smelters (see p. 358).

The total cost of treatment, running 30 stamps, with a duty of 1·37 t., is—breaker and battery, 1*s.* 10*d.*; concentration, 8*d.*; total, 2*s.* 6*d.* per ton. These figures embrace

supervision, labour, fuel, supplies, and maintenance (renewals). Taking into consideration the small number of stamps (only half what one man can attend to) and the low duty, the figures show a striking contrast to the cost at most mills. The practical results obtained during the last 4 years are shown in the annexed table :—

| | 1896 | 1897 | 1898 | 1899 |
|---|---|---|---|---|
| | oz. per ton. | oz. per ton. | oz. per ton. | oz. per ton. |
| Bullion contents . . . . . . . . | 1·583 | 1·560 | 1·540 | ·676 |
| Bullion extracted per amalgam . . | ·646 | ·494 | ·652 | ·252 |
| Bullion contained in concentrates . | ·695 | ·832 | ·636 | ·302 |
| Bullion loss in tailings . . . . | ·242 | ·234 | ·252 | ·122 |
| Bullion percental recovery . . . . | 84 | 85 | 83½ | 82 |

It will be noticed that more than half the value recovered is in the sulphurets. The tailings are carefully preserved for future treatment.

Among American examples, the great mills of the Alaskan companies take a prominent position.

At the Alaska-Mexican mill (120 head), the following figures are officially published :—

| | 1896 | 1897 | 1899 |
|---|---|---|---|
| | *s. d.* | *s. d.* | *s. d.* |
| Cost per ton milling and vanning. . . . . | 1 7¾ | 1 4 | 1 2⅓ |
| Percentage of concentrates . . . . . . . | 2·18 | 1·81 | 1·84 |
| | dwt. | dwt. | dwt. |
| Bullion recovered from amalgam per ton . . | .. | 1·432 | 1·400 |
| Bullion recovered from concentrates . . . . | .. | ·692 | ·690 |
| Bullion recovered, total . . . . . . . . | 2·417 | 2·124 | 2·090 |
| Loss in tailings . . . . . . . . . . . . | .. | ·190 | .. |
| Percental total recovery . . . . . . . . | .. | 91·8 | .. |

With wages at 10 to 14*s*. a day (plus board and lodging), even making allowance for water power during most of the time, the cost is remarkably low ; and the ratio of extraction

on such low-grade material has scarcely any parallel outside Alaska. The Treadwell Co., in 1898, with 240 stamps, had a milling and concentrating cost of 1*s.* 6½*d.*, labour amounting to 7¾*d.* and supplies to 10¾*d.*; extraction, 1·54 dwt. per ton from amalgam and ·78 dwt. from sulphurets, total 2·32 dwt.; loss in tailings not stated. In 1899–00, with 540 head, costs were reduced to 9·94*d.* per ton.

Milling and concentrating costs at the United Co.'s mill, in 1899, were 1*s.* 7½*d.* per ton; and the recoveries were 1·68 dwt. from amalgam and ·65 dwt. in sulphurets, total 2·33 dwt per ton. The percentage of concentrates was 1·93, and their average yield was 38 dwt. per ton.

Some Colorado milling and concentrating costs (1897) were:—Gold Coin, 5*s.* 8*d.*; Hidden Treasure, 4*s.* 7½*d.*; Kansas, 4*s.* 1½*d.*; other mills, 3*s.* 8*d.*

The great establishments of the Homestake Co., S. Dakota, have often been quoted as wonderful examples of expert milling, and conclusive evidence of the superiority of American principles, which may be summed up in the motto "haste and waste." When the author was in the Black Hills, in 1894, he found that the management was even ignorant of the actual contents of the ore, weighing and assaying being alike omitted. Only a rough determination of the free gold is made by a professional sampler, who pans some 50 to 55 samples a day; but of the gold which is not free, no account is taken. Until recently the tailings flowed from the plates directly into the creeks, and thousands of tons of the heavier and richer portions of these tailings choked the gulches for miles below the mills. Various estimates have been made of the proportion of sulphurets in the ore. Prof. Hofman says it "varies from 2½ and 3 % to 6 and even 10 %." The Assistant-Manager assured the author that it closely approached the last-named figure. The assay value of pure concentrates ranges from 4 to 90 dwt., and averages, according to Hofman, about 25 dwt. per ton. Some parcels the author had for experimental treatment contained 8½ dwt., and readily yielded 6½ dwt. of that (or about 70 %) to pan amalgamation. Hofman says the "total value of the ore varies from 5 to 10 dwt. per ton," or

an average of about $7\frac{1}{2}$ dwt., and "it is claimed that 85 % of the free gold is saved." The real extraction is probably not much more than 60 % of the total gold contents. In 1897, nearly 400,000 t. were milled for a return of 5·6 dwt. bullion per ton, of which 4·65 was fine gold; the concentrates gave a further ·12 dwt. Milling costs were 4*s.* $5\frac{1}{2}$*d.* per ton. In 1898, the figures were over 500,000 t., 4·5 dwt. fine gold, and ·06 dwt. from concentrates, and milling costs were 3*s.* $2\frac{1}{2}$*d.* The mills range in size from 100 to 200 stamps each. Such concentration as is attempted is done on Gilpin County bumping-tables. There being no available space for storing tailings for subsequent treatment, they are run directly into a creek to waste, and several establishments make a profit by raising and treating them. Apparently this fact has at last awakened the Company to its folly, for it is now (Sept. 1900) engaged in erecting a 1200-t. cyanide plant.

**Crushing for Concentration.**—In some few instances notably in Brazil, the proportion of coarse free gold present does not justify any means being adopted for amalgamation, and the battery is used simply as a crusher. Very coarse gold particles would in any event be arrested in the mortar box.

An interesting example of milling without amalgamation, with the sole aim of isolating the pyritic contents of the ore for concentration and chlorination, is described by Spencer Cragoe.* The ore averages a little under 9 dwt. per ton, and is "somewhat heavily pyritic, mispickel predominating," analysis giving 9·38 % iron, 4·9 arsenic, ·28 antimony, and 2·18 sulphur (some of the iron existing as magnetic oxide from *jacotinga*); and the gangue is mostly quartz, with occasional bands of soft, talcose, pyritic schist. Grizzlies set at $1\frac{1}{2}$ in. pass the ore to a Blake-Marsden breaker, thence to bins with Challenge feeders. The stamps make 85 drops of 8 in. per minute. The screens used are 35-mesh brass wire, and coupled with an 8-in. discharge, cause much sliming and limit the output. Power is generated from a 5-ft. Pelton

* Trans. Inst. M. & M., viii. 121 (1900).

approximately $\frac{1}{4}$ mile distant, and transmitted by a $\frac{5}{8}$-in. wire cable, with a loss of about 2 h.p., or 4·44 %.

There are, however, very few cases where the provision of copper-plate tables outside the battery would not result in some amount of amalgamable gold being arrested, and its recovery in that way is so simple and inexpensive that it is always worth attempting, unless the constituents of the ore prohibit the use of mercury.

**Amalgamation, Concentration, and Chlorination.**— The great majority of mills producing a valuable concentrate are content to effect as close concentration as possible and to sell the product to smelters or get it treated at outside chlorination works. But there are some who prefer, not always wisely, to conduct their own chlorination.

The Faria Co., Brazil, has been Frue-vanning the pulp from the amalgamating stamp mills, and chlorinating the concentrates, with the very meagre results of only 43·8 % extraction, the difficulty being in concentration. The yield of fine gold is 5·61 dwt. per ton, the average standard of the bullion being 892. The battery returns 4·98 dwt. bullion, and chlorination 1·35 dwt., and the total cost per ton milled is 4*s.* 9*d.* The "ore carries a good deal of clay and oxidised matter, and also some graphite, and, in stamping, produces a large quantity of slimes. At the same time, most of the gold in the ore occurs in an extremely fine state, and a large percentage of it is carried off in the slimes which escape all the various concentrating appliances." (Official Report.) It would seem in fact to be just the right sort of ore for cyanidation and just the wrong sort for concentration and chlorination.

The Ouro Petro mill returns less than 5½ dwt. per ton (81 to 82 % extraction) of gold bullion having a fineness of 870. It produces annually over 2000 t. of concentrates, having an assay value of 32·7 dwt. per ton, from which a 93 % extraction is got by chlorination. The total cost per ton milled is given at 4*s.* 10½*d.*

At the Gibraltar mill, New South Wales, in 1898, the yield in battery bullion was 14·3 dwt. per ton, and in chlori-

nation gold 4·1 dwt. per ton. The percentage of concentrates collected is about $2\frac{1}{2}$, and their average assay is $7\frac{3}{4}$ oz. per ton. Milling costs were over 4*s.* 6*d.* per ton (40 stamps) milled; and chlorination costs, 4*l.* 9*s.* 8*d.* per ton treated. Both are very high. Disposal of the concentrates to smelters has since been adopted. Total cost, including cyanidation of a portion of the tailings, is given at 6*s.* 11*d.* per ton milled.

The Alaskan mills, with all their advantages in large product and consequent scale of operations, after years of experience in chlorinating their own concentrates, have recently abandoned the practice, and now send their sulphurets to a distant smelter. The cost of labour for concentration, at the Treadwell mill, in 1898, was 6*s.* $11\frac{1}{2}$*d.* per ton of concentrates, and other charges brought the total to about 16*s.* 8*d.*; chlorination cost 32*s.* $7\frac{1}{2}$*d.* per ton of concentrates.

**Chlorination only.**—Chlorination mills and the results and costs achieved by them have been already fully described (pp. 410–72). Except for a new impetus given to the process by the needs of the Colorado tellurides, it seems to be on the decline, the smelters competing with it for rich ore and concentrates, and cyanidation being more cheaply applied to poor ores.

The custom mill of the Delano Co., at Boulder, Colorado, built for dealing with the Cripple Creek tellurides, is shown in Fig. 201. The ore is delivered by road waggons to a series of storage bins *a*, having a capacity of 25 t. each, to suit small parcels. Thence it is wheeled and hand-fed to the breakers *b*, and passes through the automatic samplers *c*, and rotary drier *d*, to the fine-crushing rolls *e*. These last are in two sets, 14 × 30 in., making 45 rev. a minute, and reducing the ore to pass two 20-mesh (No. 24 wire) screens of 70 sq. ft. area each. From the screen-bins *f*, a spiral conveyor *g*, 9-in. diam., 8-in. pitch, at 25 rev. a minute, transfers the powdered ore to an endless belt, 8 in. wide, 7-ply, with pure rubber flanges $\frac{5}{8}$ in. high, travelling at 100 ft. per minute, which discharges into the feed hopper *h* of the Pearce furnace *i*. The roasted ore passes from the furnace through a set of 120 cool-

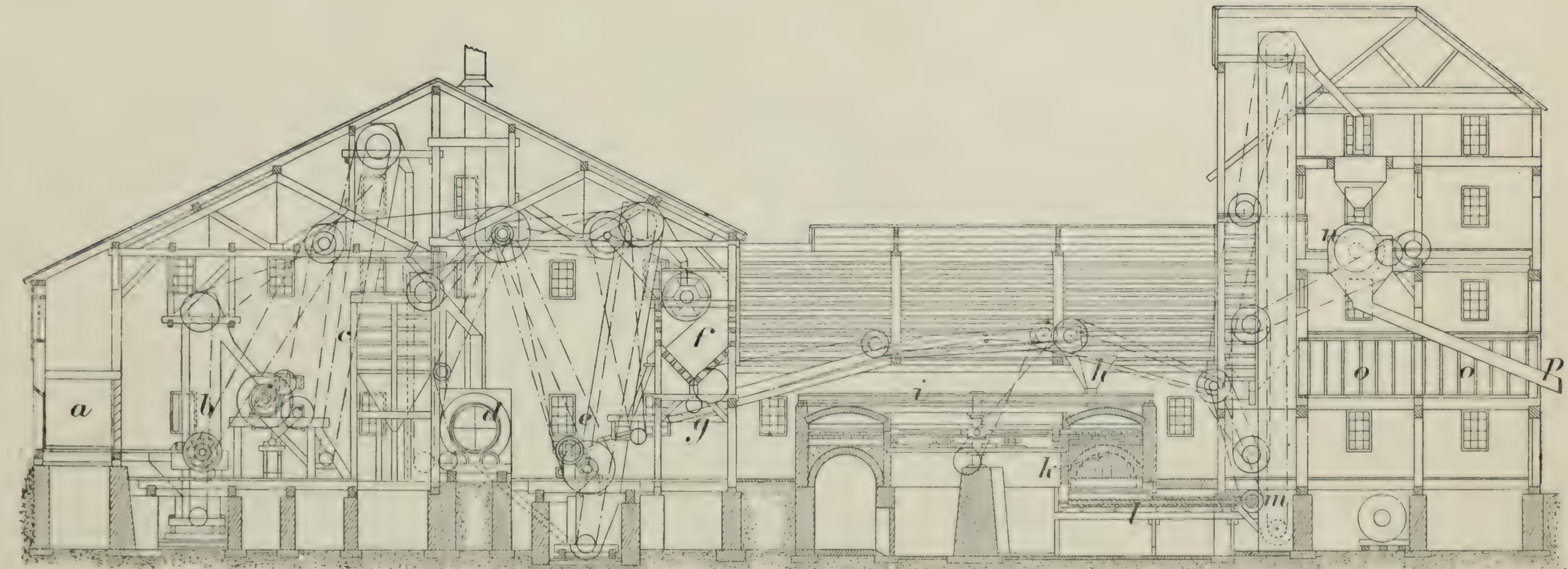



Fig. 201.—A Chlorination Mill.

ing tubes *k*, 4 ft. × 3 in., immersed in cold water, the passage occupying 20 minutes, and reaches a spiral conveyor *l*, which empties into the boot of elevator *m*. The spout from the elevator feeds two 35-t. bins, which supply the 5-t. steel hoppers of the 3 chlorinating barrels *n*. These measure 8 × 5 ft., and are fitted with inside filter of screened sand and perforated sheet lead. The gold liquors are drawn off into a series of 8 tanks *o*, being passed from one to another twice in order to effect clarification, a monte-jus or pneumatic pressure being employed for the purpose. The tanks have close lids of ¾-in. tongued and grooved boards, to prevent escape of gas into the mill; and above them runs a 12 × 18-in. wooden flue connecting with each tank by an 8 × 8-in. branch, and with a 40-in. fan running at 750 rev. a minute, discharging all noxious fumes at a good height in the open air. Leaching time averages 2½ hours per charge; and consumption of chemicals is about 10 lb. bleaching powder and 20 lb. sulphuric acid per ton. The latter is run from the railway tank-cars into a steel sump tank, and is raised thence as required by compressed air. Precipitation is effected by $H_2S$. Solid residues are discharged outside the mill by shoots *p*. The capacity of the plant is 50 t. per 24 hours, and the following power is provided:—a 60-in. × 16-ft. horizontal return tubular boiler (1050 sq. ft. heating surface); a 14 × 20-in. horizontal engine, making 130 rev. a minute, for milling plant; an 8 × 8 × 8-in. air compressor for raising liquors and acid; a 7 × 10-in. high-speed engine and a 7-kilowatt generator for electric lighting; and a 10 × 12-in. engine for driving furnace and barrels. The fuel used in furnace is short-flamed lignite costing 6*s*. 8*d*. per ton at the mill, in preference to semi-bituminous coal at 14*s*. 7*d*.; but the roasting is not very efficient when full duty is required of the furnace, the sulphur left in amounting then to ·4 %. On 15-dwt. ore the extraction averages 92 % (13·8 dwt.), leaving 1·2 dwt. loss; sometimes the residues do not exceed ·7 dwt. Working costs are given by J. Roger * as follows:—Labour, 4*s*. 2*d*.; chemicals, 2*s*. 3½*d*.; fuel, 2*s*. 1*d*.; oils, etc., 3*d*.; water, 2*d*.; wear and tear, 1*s*. 0½*d*.;

* Engineering, July 22, 1898.

incidentals, 7*d.*; total, 10*s.* 7*d.* per ton. These figures are based on bleaching powder at 10*s.* 5*d.* and sulphuric acid 5*s.* 5*d.* per 100 lb. at the mill.

Another modern mill for treating Cripple Creek ores is that of the El Paso Co., at Florence, Colorado, illustrated in Fig. 202. A special feature of it is that the fuel used both for generating steam and for roasting is petroleum residue, which is delivered in tank-cars from the refinery about a mile distant, and is pumped into a high-level store-tank whence it can gravitate to any part of the works. This store-tank contains a small exhaust-steam coil, and each supply-pipe has a steam-pipe beside it under the same covering, to maintain the necessary fluidity. The ore is brought in on a high-level tram-line, and slides automatically down grizzlies to the mouths of the breakers. The coarsely broken ore is brought on elevated tracks *a* to a bedding-floor *b*, which is in two sections, one being filled while the other is being emptied. Each section, when covered to a depth of 6 ft., holds about 600 t. of ore; in this way fairly uniform value and character can be ensured, which would be impossible otherwise. From the bed, the ore is taken back for drying and fine crushing by a 12-in. rubber-flanged belt conveyor running at 100 ft. a minute, with arrangements for distributing. The rotary driers deliver to an elevator boot, and the dried ore is raised to screens set over the fine rolls. These are 14 × 30 in. and make 45 rev. a minute. Two Pearce furnaces *c*, 40 × 8 ft., do the roasting, Reed burners being used. Sulphur is reduced from $1\frac{1}{2}$ or 2 % to ·1 or ·15 %. The furnace gases traverse a sheet-iron flue *d*, which leads to a large brick dust-chamber. The roasted ore goes through an automatic cooler *e* (since abandoned in favour of hand-spreading on a cooling-floor) and by a scraper conveyor *f* to an elevator *g*, and is fed into the hoppers of 8 chlorinating barrels *h* of the same pattern and size as at the Delano mill. The barrels are driven by flat-face paper friction-gears keyed to line-shafting. Tightly-covered liquor-tanks *i*, 8 in number, with a total capacity of 100,000 gal., receive the gold solution; they are ventilated as at Delano. The liquors pass through launders *k* carrying steam-coils

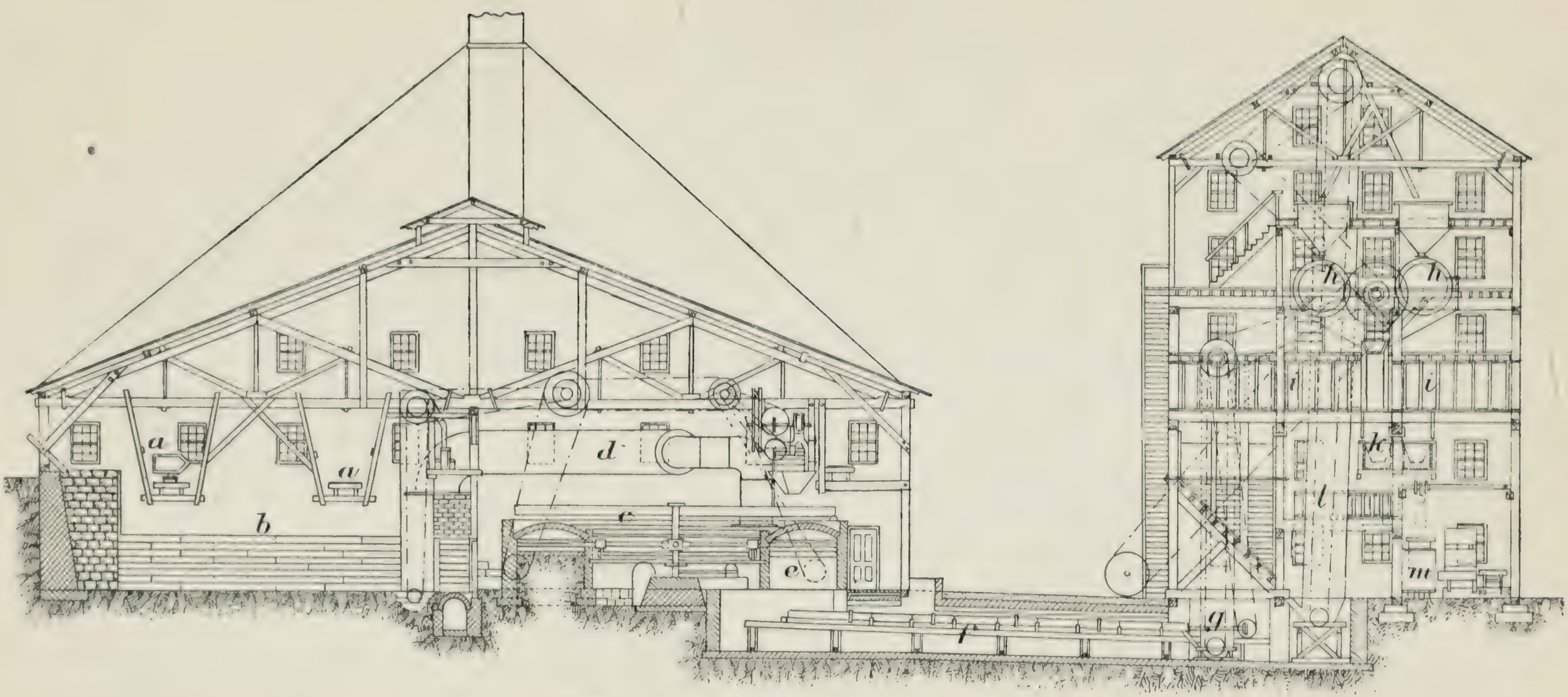



FIG. 202.—A CHLORINATION MILL.

where they are heated to about 120° F., and thence through sand-beds in lead-lined tanks *l* for clarification. Precipitation is done by $\frac{1}{4}$-in. charcoal in lead-lined steel boxes *m* 40 in. diam. and 5 ft. deep, which are conveyed by a hand-power travelling crane to the burning room, where their contents are emptied into $\frac{7}{8}$-in. cast-iron pans 5 × 3½ ft. × 7 in., and burned off slowly by bottom heat. The approximate cost given by J. Roger,* on a basis of bleaching powder at 10*s.* and sulphuric acid at 5*s.* 2½*d.* per 100 lb. at the mill, is as follows:—Labour, 3*s.* 9*d.*; chemicals, 2*s.* 2½*d.*; fuel, 1*s.* 10½*d.*; water, 2½*d.*; oils, etc., 2½*d.*; wear and tear, 1*s.* 3*d.*; incidentals, 7½*d.*; total, 10*s.* 1½*d.*

For roofing chlorination mills and roasting works, galvanised iron is not suited, it being very quickly destroyed by the acid fumes. In some localities slates or tiles may be used with advantage, being much more durable and much less costly, besides permitting a style of fixing which affords abundant opportunity for escape of noxious gases. In other cases, cost of transportation would put such heavy materials out of the question. Then nothing is better than the article known as "asbestic," which is prepared from short-fibred asbestos, at Danville, Quebec. It is a kind of felt,—light, fireproof, waterproof, and acidproof,—made in "squares" of 108 sq. ft., and varying in thickness up to 50 lb. per square. Its cost is approximately 10 to 15*s.* per square. It is easily fixed.

**Amalgamation, Chlorination, and Cyanidation.**—Here and there is to be found a mill which, after the extraction of free gold by amalgamation, and before passing the tailings on to cyanide treatment, interposes concentrating appliances and eliminates from the tailings a small proportion of relatively rich material suitable for chlorination. A good example of this procedure is the Robinson mill, Transvaal, which buys and treats the vanner product of the Ferreira and other adjacent mills. In 1899, milling and vanning costs were returned at 3*s.* 2*d.* a ton, and chlorinating and cyanidating (in one item) at 2*s.* 5½*d.*; total 5*s.* 7½*d.* The Ferreira concentrates treated

* Engineering, August 19, 1898.

by the Robinson Co. were 2½ % and assayed just over 5 oz per ton; the extraction returned was 96 %, amounting to 4·882 oz. fine gold per ton treated, or 2·407 dwt. per ton milled. The percental recoveries of the Ferreira Co. were: amalgamation, 57; concentration (chlorination), 11; cyanidation, 24; total, 92 %.

**Amalgamation and Cyanidation.** — By far the most usual combination of processes is amalgamation of the free coarse gold followed by cyanidation of the tailings, the latter being usually separated into sands and slimes for distinct treatment. This separation often involves more or less concentration by simple automatic means, the concentrates being also subjected to cyanide.

*Mills.*—In giving a couple of examples of amalgamation-cyanidation mills, choice has been made of one using battery-amalgamation and one using rolls (dry) and pans.

The former (Figs. 203, 204) is the new custom mill of the Montana and Denver Reduction Co., at Bearmouth, Montana, built by the E. P. Allis Co. There are 5 receiving bins *a*, each of 20-t. capacity, delivering to a small truck which feeds a 9 × 15-in. Blake breaker *b*. Hence it is raised by an elevator and dumped into another car running on a track at the back of the battery-bins *c*. A certain amount of classification is carried out, the oxidised ore being stamped to 40 mesh for the sake of its free gold, while pyritic ore is milled only to 30 mesh in view of concentration. The battery is provided with outside plates *e*. The pulp passes hydraulic sizers, so that 4 grades are delivered, each going to a separate double-decked Wilfley table *f*. The tailings finally enter into four 100-t. leaching tanks *i*, the liquors from which are drawn off by means of vacuum-tank *m* into intermediate tanks *l*, flow through precipitators *n* to sumps *o*, and are raised thence by pumps *p* into standardising tanks *k*. The natural gradient of the site, about 30°, has been utilised in arranging the plant.

The mill shown in Fig. 205, by the same builders, has a swing-jaw breaker set over bin delivering by belt conveyor to a rotary drier, which feeds the crushing rolls. These discharge

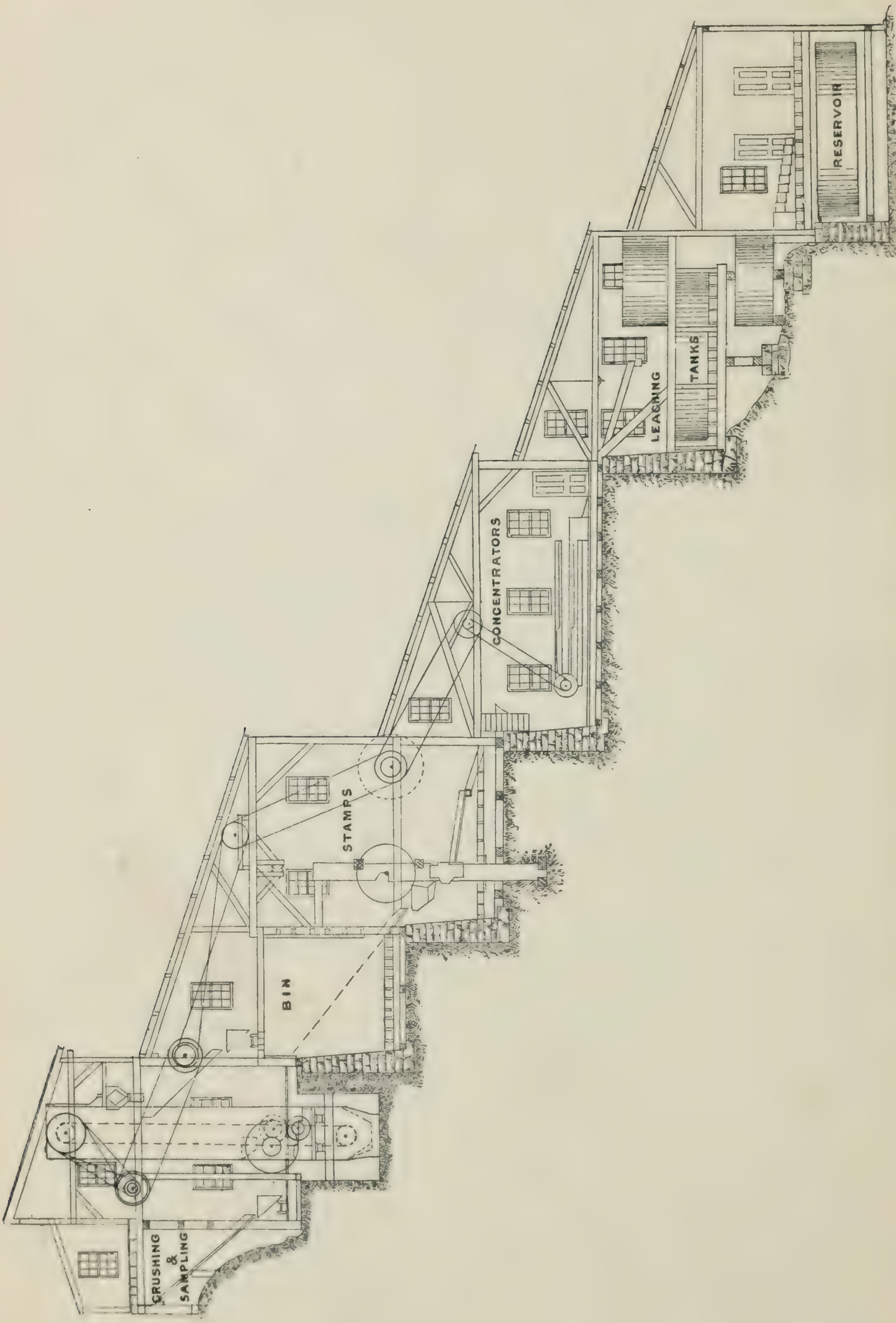



Fig. 203.—Amalgamation and Cyanidation Mill.

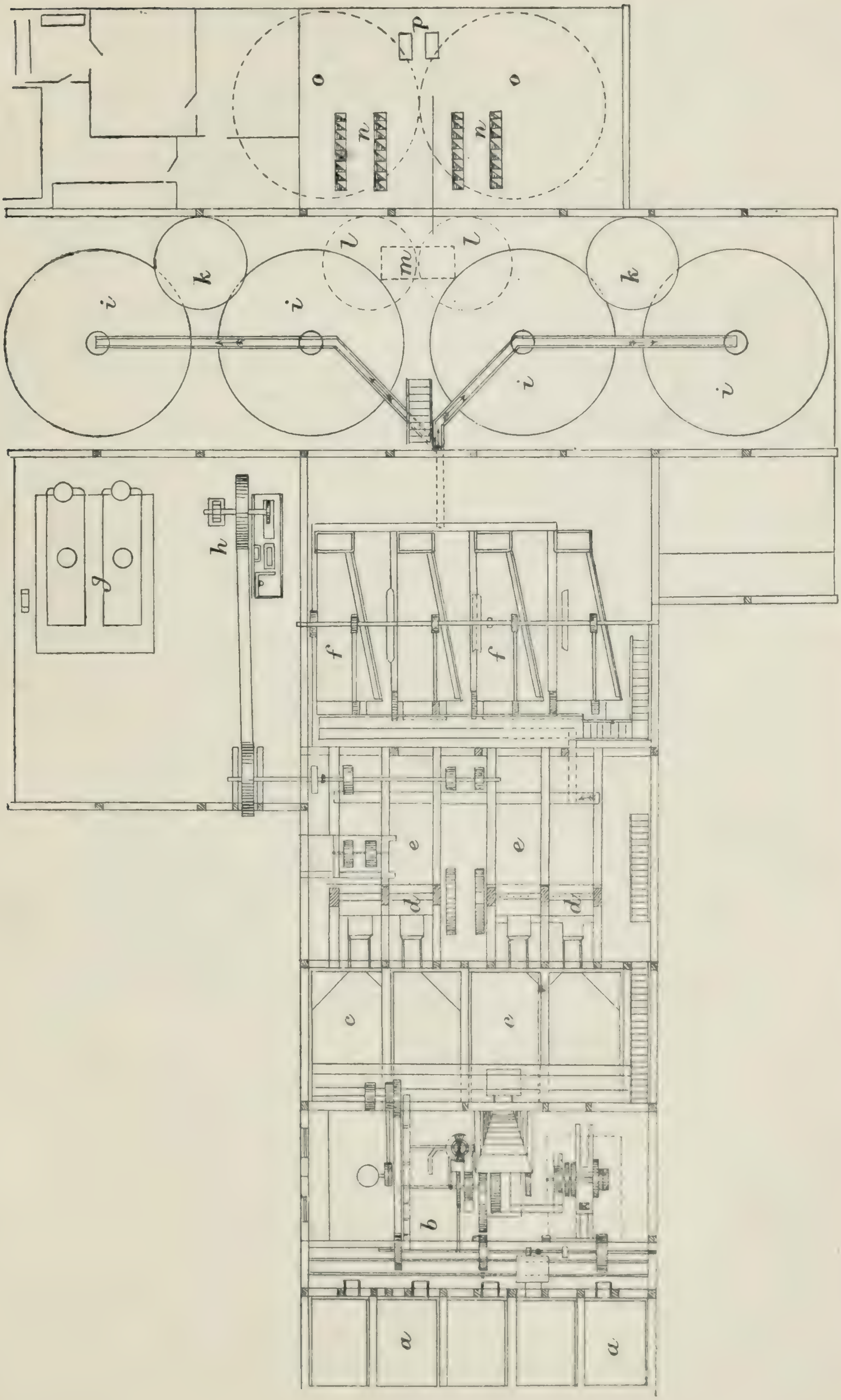



FIG. 204.—AMALGAMATION AND CYANIDATION MILL.

to amalgamating pans, and the issue of tailings is into two tiers of leaching tanks for cyanidation.

*Methods and Results.*—A Brazilian example exists in the Morro Velho (100-stamp) mill, where, for some occult reason, much mystery surrounds the second treatment, which is called the "oxygen process." It would seem to be simply cyanida-

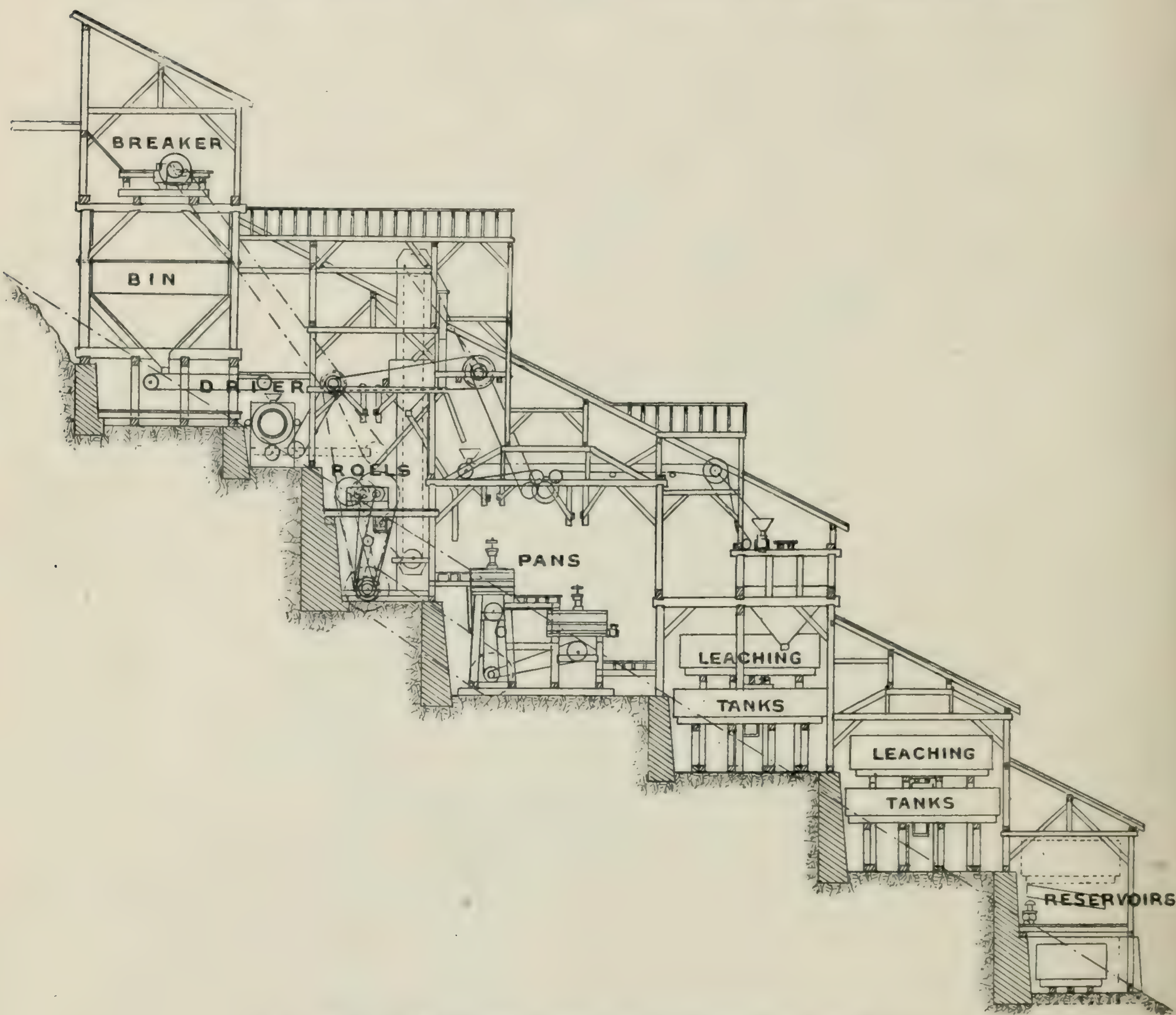



FIG. 205.—AMALGAMATION AND CYANIDATION MILL.

tion with the aid of an oxidising agent or oxygen-carrier to take the place of the bromine used in Sulman's method and in the so-called Diehl process. Official returns convey but the flimsiest information concerning results attained and costs incurred, but it is stated in a recent one that the recovery by amalgamation amounts to 73·12 % (16·14 dwt.) of the contained bullion, and, by the oxygen process, 12·93 (2·86 dwt.);

total, 86·05 % (19 dwt.). The total cost is given at 7*s*. per ton.

A very interesting case occurs at Deloro, Canada, where for many years chlorination of the mispickel ores was regarded as imperative. Then bromo-cyaniding displaced chlorination, and recently (1898) it has been found quite feasible to extract 60 % of the value, or 7 out of 13 dwt., by amalgamation. The new plant is working on these lines, 20 stamps treating the ore as ordinary free milling; the arsenical pyrites are then saved in vanners for bromo-cyaniding (and to be finally roasted for their arsenic), while the effluent slimes are so low-grade as not to be worth treatment of any kind. This course is giving a maximum extraction at a minimum cost.

The Indian mines, after a long reliance upon battery amalgamation and pan-grinding of tailings, have also come to recognise the merits of cyanide. Thus the Coromandel Co., in 1899, extracted 3·73 dwt. bullion per ton from amalgam and ·67 dwt. by cyaniding the tailings, or a total of 4·4 dwt., which, however, is not calculated to pay dividends.

At the Mysore mill, in 1896, the costs were: milling, 5*s*. 0¾*d*.; pan amalgamation of tailings, 6*s*. 10*d*.; cyanidation of tailings, 3*s*. 11*d*.; or a total cost per ton treated of 15*s*. 9¾*d*. The extractions, in 1898, were stated as follows: by battery amalgamation, 29½ dwt.; by pan amalgamation of tailings, 4 dwt.; by cyanidation of sands, 2·35 dwt.; and by similar treatment of slimes, 1·18 dwt.; total, 37·03 dwt. per ton.

But it is in the Transvaal that the real development of this combination of methods has been witnessed, and here it is practically universal. According to W. Y. Campbell* the average cost of milling, "from the time the ore comes to daylight till it leaves the mill for the leaching works as sands" is 3*s*. 6*d*., while some exceed that. For a typical 80-stamp mill, with a duty of 4½ t., he would itemise the cost thus: white labour, 13¼*d*.; black labour, 5¾*d*.; water, 1*d*.; stores, 6¾*d*.; fuel, 8¼*d*.; sundry, 6*d*.; total, 3*s*. 5*d*. per ton. But while the ton is supposed to be 2000 lb., it is in many instances not

* En. & Min. Jl., July 24, 1897.

more than 1500 lb., and Campbell puts the average Rand ton at 1750 lb., which means that the actual cost is 12½ % higher. As to extraction ratios, he estimates that in well-run mills 65 % of the assay value is recovered as amalgam, while in many cases it ranges between 45 and 50, and he would put the average at 55 %. The melted bullion contains 9 to 11 % silver, and the fineness stands at 880 to 890.

In the tables on pp. 749, 750, are given such figures as have been procurable in reference to S. African mills, based upon their own official statements. Notwithstanding a lack of absolute reliability, inasmuch as no weighing of ore is done, and many of the estimated tonnages are decidedly elastic, the figures are full of interest. Bearing in mind the scale of operations, the remarkable adaptability of the ore to simple amalgamation and cyanidation, and other unusually favourable conditions, it must be admitted that, except in a few isolated cases, the total extraction is not what it might be and the working costs are inordinately high.

The most prominent installation in the United States is the Standard mill, at Bodie, California. The extraction (1896) from ore assaying 19·02 dwt. gold was about 13 dwt. or 65 % by amalgamation, the tailings assaying a little over 6 dwt. Cyanide recovers 70 to 80 % of this. The total cost is given as 13*s.* per ton, of which less than half is for cyanidation. According to Bosqui * "there is probably no locality where the simple cyaniding of tailings is carried on with such satisfactory results, there being nothing in the ore to complicate the process."

From the official reports of the De Lamar Co., Idaho, for 1897 and 1898, several interesting points may be noted. In the former year, 40 stamps, with a duty of 3·48 t. per 24 hours, milled 40,453 t. assaying 14 dwt. gold per ton, and returned a total of just over 71 % of the value, at an aggregate cost of 17*s.* 2½*d.* per ton. The treatment embraced stamping, pan amalgamation, and cyanidation by Pelatan-Clerici method. Of total costs, labour amounted to 6*s.* 11½*d.*; fuel, 4*s.* 1*d.*; mercury, 3*s.*; maintenance, 1*s.* 10*d.*; and other

* 'Cyanide Process,' p. 124 (1899).

TRANSVAAL MILLING RETURNS.

| Mill. | Date. | Ore. | Recovered by amalgam. | | Recovered by cyanide. | | | Total recovery. | | Untreated. | | Resi-dues. | Total loss. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Assay dwt. | dwt. | % | % treated | dwt. | % | dwt. | % | % | Assay dwt. | Assay dwt. | dwt. |
| Angelo | '99 | 17·48 | 8·46 | 48 | 79 | 4·95 | 28 | 13·42 | 76 | 17½ | 5·80 | 1·61 | 4·06 |
| Bonanza | '99 | 26·77 | 16·01 | 60 | 99½ | 8·93 | 33 | 24·94 | 93 | .. | .. | .. | 1·83 |
| Crown Deep | '98 | 11·21 | 5·36 | 48 | 82½ | 4·44 | 39½ | 9·80 | 87½ | 17½ | .. | .. | 1·41 |
| Crown Reef | '99 | .. | 9·32 | .. | 98 | 5·67 | .. | 14·99 | .. | .. | .. | .. | .. |
| Driefontein | '99 | 12·16 | 5·28 | 43½ | 81 | 4·20 | 34½ | 9·48 | 78 | 19 | 7·22 | 1·46 | 2·68 |
| Durban Roodepoort | '98 | 12·85 | 7·45 | 58 | .. | 3·34 | 26 | 10·79 | 84 | .. | .. | .. | 1·02 |
| Ferreira | '98 | 21·93 | 12·51 | 57 | 98½ | 7·68 | 35 | 20·19 | 92 | 1½ | .. | .. | .. |
| Geldenhuis | '98 | .. | 6·46 | .. | 98½ | 4·45 | | 10·91 | .. | 1½ | .. | .. | .. |
| Geldenhuis Deep | '98 | 11·40 | 6·21 | 54½ | 100 | 3·34 | 29¼ | 9·55 | 83¾ | .. | .. | .. | 1·85 |
| Glen Deep | '98 | 11·96 | 6·35 | 53½ | .. | 4·13 | 34½ | 10·50 | 87¾ | .. | .. | .. | 1·46 |
| Jumpers | '99 | 10·90 | 5·78 | 53 | 69 | 2·73 | 25 | 8·51 | 78 | 31 | .. | 2·39 | .. |
| Jumpers Deep | '99 | 13·31 | 6·78 | 51 | 69 | 3·85 | 29 | 10·63 | 80 | 31 | .. | .. | 2·68 |
| Langlaagte Deep | '99 | 9·93 | 5·30 | 53½ | .. | 2·59 | 2½ | 7·89 | 56 | .. | .. | .. | .. |
| New Comet | '99 | 10·84 | 5·22 | 48¼ | 82½ | 3·26 | 30 | 8·48 | 78¼ | 17½ | 6·15 | .. | 2·36 |
| Nourse Deep | '99 | 11·34 | 5·74 | 50½ | 100 | 4·46 | 39½ | 10·20 | 90 | .. | .. | .. | 1·14 |
| Paarl | '98 | 8·36 | 4·30 | 51½ | .. | 2·25 | 27 | 6·55 | 78¼ | .. | 3·44 | 1·13 | 1·81 |
| Robinson | '97 | 18·82 | 11·45 | 60¾ | .. | 5·36 | 28½ | 16·80 | 89¼ | .. | .. | .. | .. |
| Do. | '98 | 19·37 | 11·56 | 59¾ | .. | 5·95 | 30¾ | 17·53 | 90½ | .. | .. | .. | .. |
| Rose Deep | '98 | 11·99 | 6·23 | 52 | .. | 4·19 | 35 | 10·42 | 87 | .. | .. | .. | .. |
| Sheba | '98 | .. | 5·33 | .. | .. | 4·92 | .. | 10·25*a* | .. | .. | .. | .. | .. |
| Simmer and Jack | '97 | .. | 5·92 | .. | .. | 5·77 | .. | 11·69*a* | .. | .. | .. | .. | .. |
| Witwatersrand | '98 | 9·80 | 5·37 | 55 | 70 | 1·88 | 19 | 7·26 | 74 | .. | .. | .. | .. |

*a* Bullion.

TRANSVAAL MILLING COSTS.

| Mill. | Date. | Stps. | Duty. | Costs. | | | | |
|---|---|---|---|---|---|---|---|---|
| | | | | Breaking. | Milling. | Concent. | Cyaniding. | Total. |
| | | | | s. d. | s. d. | s. d. | s. d. | s. d. |
| Angelo | '99 | 80 | 4·56 | 1 11¼ | 3 0 | .. | 2 10 | 7 9¼ |
| Bonanza | '99 | 40 | 4·89 | 2 0 | 4 7 | .. | 6 7½ | 13 2½ |
| Crown Deep | '98 | 160 | 4·97 | .. | 3 7 | .. | 2 10 | 6 5 |
| Crown Reef | '99 | .. | .. | 0 11¼ | 2 6¾ | .. | 3 0½ | 6 6½ |
| Driefontein | '99 | 110 | 4·66 | 2 2¼ | 3 1¼ | .. | 2 7 | 7 10½ |
| Durban Roodepoort | '98 | 80 | 4·51 | .. | 3 2½ | .. | 2 1½ | 5 4 |
| Durban Roo. Deep | '98 | 50 | 4·90 | .. | 4 9½ | | 3 1½ | 7 11 |
| Ferreira | '98 | 80 | 4·73 | 1 2½ | 2 8 | 0 11½ | 4 0 | 8 10 |
| French Rand | '98 | 60 | .. | 1 2¾ | 2 5½ | .. | 3 3½ | 6 11¾ |
| Geldenhuis | '98 | 120 | 5·02 | 0 7¾ | 2 2½ | .. | 2 8 | 5 6¼ |
| Geldenhuis Deep | '98 | 190 | 4·52 | .. | 2 11 | .. | 2 5½ | 5 4½ |
| Glen Deep | '98 | 100 | 5·16 | .. | 3 10½ | .. | 2 8¾ | 6 7¼ |
| Glynn's Lydenburg | '98 | 20 | 4·19 | .. | 2 0 | .. | 4 6½ | 6 6½ |
| Johannesburg Pioneer | .. | .. | .. | .. | 5 5 | .. | 3 5½ | 8 10½ |
| Jumpers | '99 | 100 | 4·27 | 0 11½ | 4 6 | .. | 2 4 | 7 9½ |
| Jumpers Deep | '98 | 85 | 4·92 | .. | 4 4½ | .. | 2 1 | 6 5½ |
| Langlaagte Deep | '99 | 100 | 5·54 | .. | 4 4 | .. | 2 10¼ | 7 2¼ |
| New Comet | '99 | 90 | 4·38 | 1 7¾ | 3 0¼ | .. | 2 7½ | 7 3½ |
| New Goch | '98 | 60 | .. | .. | 4 9 | .. | 2 6¾ | 7 3¾ |
| Nourse Deep | '99 | 100 | 4·81 | .. | 3 5½ | .. | 3 5¾ | 6 11¼ |
| Paarl | '98 | 60 | 4·19 | 1 0 | 2 9½ | .. | 2 6½ | 6 4 |
| Robinson | '98 | 140 | 4·40 | 0 6¾ | 2 4¾ | 0 4¾ | 3 0 | 6 4¼ |
| Rose Deep | '98 | 150 | 4·90 | .. | 2 9½ | .. | 3 2½ | 6 0 |
| Sheba | '98 | 200 | .. | .. | 5 8¾ | 0 10 | 4 7 | 11 1¾ |
| Simmer and Jack | '97 | 280 | 4·80 | .. | 3 7 | 0 3 | 2 5½ | 6 3½ |
| Transvaal Estates | '98 | 60 | .. | .. | 1 10 | .. | 4 3 | 6 1 |
| Van Ryn | '98 | 160 | .. | .. | 3 0 | .. | 1 7½ | 4 7½ |
| Village | '98 | 100 | .. | 0 6½ | 2 8¾ | .. | 2 9 | 6 0½ |
| Windsor | .. | 50 | .. | 0 10¾ | 4 0 | .. | 3 1 | 7 11¾ |
| Witwatersrand | .. | .. | .. | 0 10¼ | 2 2 | .. | 2 0½ | 5 0¾ |

chemicals, 1*s.* 4*d.* per ton. In 1898, pan amalgamation and Pelatan-Clerici were discarded as too costly, and a percolation plant was substituted, which is calculated to reduce milling expenses by about 8*s.* a ton. The comparative costs and extractions by the three methods are thus given :—

| | Cost per ton. | Extraction % |
|---|---|---|
| | *s.* *d.* | |
| Pan amalgamation . . . . . | 17 3½ | 73 |
| Pelatan-Clerici . . . . . . . | 13 8½ | 75 |
| KCy leaching . . . . . . . | 12 0 | 50 |

"The low percentage saved in the leaching vats was due to leakage and losses," assay results showing 65 %. There is certainly good room for improvement both in costs and extraction. The average treatment cost for 1898 is given at 16*s.* 2*d.* per ton.

The Montana Co. in 1897, running 50 stamps, got an extraction of 11¾ dwt. bullion per ton, gold being responsible for 90 % of the value. The process was stamp milling and pan amalgamation. In 1898, ore suited to the latter treatment became exhausted, and plates and vanners were relied on, with subsequent cyanidation of tailings. The average total yield was 6·94 dwt. Of this, 2·31 dwt. came from tailings, at a cost of 4*s.* 10½*d.* per ton.

Several Washington mills, such as the Republic and the Mountain Lion, the former of which adopted and has now abandoned pan amalgamation and Pelatan-Clerici treatment, are going back to stamp milling and plate amalgamation for the coarse gold, and plain cyanidation for the remainder. At the latter mill, stamps will crush to 30 mesh, and the pulp, after passing the plates, will be re-ground to 60 or 80 mesh in Huntington mills. Thence it will flow to agitating tanks, and be treated for 7 hours with ·5 % KCy solution. The tanks will be of 30 t. capacity, and will have agitators of the propeller type. A second treatment with ·25 % liquor will be given in filtering tanks 24 ft. diam. and 8 ft. deep, filtra-

tion occupying 24 hours. Total milling cost is expected to be about 9 to 10*s*. a ton.*

West Australian practice is full of contradictions, and may still be regarded as in a somewhat indecisive condition, notwithstanding that several mills have been built at extravagant cost on various lines, some, it must be admitted, prematurely.

While there are here, as on most other new fields, two classes of ore—the superficial and oxidised on the one hand, and the deeper-seated pyritic on the other—the former will soon disappear, and in any event its treatment does not present great problems. The complex unoxidised product of the lower levels must be regarded as the main proposition, and this is further complicated by the nature of the gangue. The ore is essentially gold-impregnated basic rock, containing much alkali and alkaline earth, some coarse free gold, and various telluride minerals in addition to ordinary sulphides. The basic gangue is prone to create an undue proportion of slimes when wet-milled, and prohibits chlorination. The fine gold compels a leaching process for its recovery, which is not adapted to saving the heavy gold. Dry milling is desirable to avoid slimes, but it is more costly than wet, and provides no means of arresting the coarse gold, while in a stamp battery this is automatically done. The pyritic contents hinder the progress and enhance the cost of leaching methods, necessitating either a preliminary roast or the use of special aids (such as bromo-cyanogen), which are also expensive. Concentration is not effective in many cases, owing to the extreme fineness of the gold particles ; besides, efficient concentration has not yet been attained by dry methods.

Then again the questions of fuel and water supply are further factors of great importance. The present sources of fuel are eucalyptus and mulga, drawn from the "bush" which prevails in many regions, but it is being rapidly destroyed, and railway extension will be necessary to maintain the supply ; its price now ranges between 20 and 30*s*. a cord. The best of it is about equal to oak for steam raising.

† En. & Min. Jl., March 10, 1900.

Coal awaits transportation. All the local water is more or less impregnated with alkaline salts, and but few mills possess even of that sufficient for their needs, and are compelled to purchase, the rates varying from 3 to 30*s.* per 1000 gal. Water consumption is calculated to vary between 250 and 400 gal. per ton of ore treated, and its cost between 1*s.* and 12*s.* 6*d.* per ton.

Under these circumstances, stamp milling has cost about 6*s.* per ton in the few cases where it has been tried. Roasting is calculated by H. C. Hoover * as not likely to exceed 10*s.* per ton, but judging from the great variety of furnaces under trial it would seem that an altogether satisfactory issue to roasting experiments has not yet been obtained. The authority quoted above anticipates the total cost of treating telluride ores at Kalgurli to be between 35 and 45*s.* a ton, while some free-milling ores may be amalgamated and cyanidated for 28 to 36*s.* per ton. These latter would contain a value of about 10 dwt. gold. The richer telluride ore ($3\frac{1}{2}$ to $4\frac{1}{2}$ oz.) is being handpicked and shipped to distant smelters, much as is done with a similar article in Colorado, though transport and treatment charges amount to 5*l.* a ton.

A difficulty introduced by roasting is that the tellurous acid generated by decomposition of the tellurides at once combines with the ferric oxide derived from the pyrites to form a new compound, the tellurite of iron, and this has a tendency to envelop the larger grains of free gold, giving them a skin which is quite impervious to mercury, so that they escape amalgamation in the ordinary way. This has favoured the adoption of pan amalgamation for the coarse gold, the rubbing there encountered having a useful effect in destroying the tellurite of iron scale more or less. Indeed it was experimentally found by A. C. Claudet † that pan amalgamation (with ordinary water as distinguished from the densely saline local article) would extract 77 % of the total gold contained in $3\frac{1}{4}$-oz. ore, and of the $3\frac{3}{4}$ oz. carried by the finest pulp (amounting to $58\frac{1}{2}$ % and passing 120 mesh) he

* En. & Min. Jl., Dec. 17, 1898.

† Trans. Inst. M. & M., v. 329 (1897).

recovered in this way $3\frac{1}{2}$ oz. or over 93 %. While this may not point to a practical method of treatment on the working scale, it at least is instructive as showing that the gold is present in a free and uncombined condition, though extreme comminution of the ore may be needed to liberate it sufficiently.

The methods in use have practically resolved themselves in all cases into a combination of amalgamation and cyanidation. The variation has been in the sequence of these two operations and in the adoption or rejection of roasting. Some favour wet milling and plate amalgamation on raw ores, using bromo-cyanidation for the tailings; others go for dry milling, roasting and simple cyanidation, with pan amalgamation for the heavy gold as a subsequent stage. Each has its good and its bad points, and opinions are divided with considerable emphasis. It is not too much to say that if roasting could be modified so as to occupy less time, involve less plant, and cost materially less per ton, the benefits conferred by it would probably cause it to be universally adopted. Possibly it would also permit a limitation of the crushing, producing less slime to interfere with leaching and giving the plant an increased output for the same power, or an equal output for less power. Should the method briefly mentioned on page 366, and now on its trial, prove to be as applicable to tellurides as to ordinary sulphides, it may be expected to definitely settle the Kalgurli problem in favour of dry milling and simple cyanidation, with the advantage of diminishing the proportion of product requiring filter-press leaching, if not avoiding it entirely. It is an open question whether automatic concentration of the cyanidation residues would not be as effectual as pan amalgamation in arresting unattacked particles of gold, and at much less cost. Indeed it may be that the temperature created in the roasting method alluded to would be so low as to avoid the reaction between tellurous acid and ferric oxide which goes so far towards preserving the coarse gold against attack.

On the other hand, a successful consummation of the Coolgardie freshwater scheme may give an impetus to wet

crushing; or it may result in a sulphating roast only being attempted, followed by washing out soluble salts as a preliminary to cyanidation.

The great mass of 5- to 15-dwt. ore cannot stand the expense of present methods, and only that of more than 15-dwt. value is now being dealt with. From this the extraction is reported to be 85 to 90 % at a cost of about 19*s.* a ton.

Actual figures are published in only a few instances.

The North Boulder Co., in 1898, operating on oxidised ore estimated at 23¾ dwt. per ton, reported an extraction of 17¼ dwt. or 72¾ %, the tailings assaying 6½ dwt.

At the Ivanhoe, in 1899, the 60-stamp mill treated over 60,000 t. of oxidised ore estimated to carry 30·7 dwt. fine gold per ton. The pulp from the battery passes over 50 ft. of canvas strakes (which eliminate 2 % sulphurets), and is then classified into 50 % sands and 48 % slimes. The concentrates assay about 12 oz. per ton, but have not been treated yet.

In 1899, the Lake View Consols dealt with nearly 100,000 t. of oxidised ore in the battery. Milling cost, 5*s.* 4¾*d.*; cyanidation of sands, 4*s.* 7*d.*; slimes treatment, 6*s.* 8½*d.*; total, 16*s.* 8¼*d.* This company is about to commence sulphide treatment on the large scale by bromo-cyanogen without roasting; dry crushing and roasting have been tried experimentally, but the battery and bromo-cyanide have triumphed. Sands and slimes will be separated as usual, and filter-presses will be availed of for the latter.

In New Zealand, the Talisman Co. has been one of the first to recognise the importance of amalgamation for saving the coarse gold, although dry crushing has been followed (stamps and ball mills). The respective gold extractions in 1899 were: by amalgamation, 1·94 dwt. (12·1 %); by cyanidation, 13·01 dwt. (81 %); total, 14·95 dwt. (93·1 %); loss, 1·12 dwt. (6·9 %); and for silver: 1·49 dwt. (1·1 %) and 33·03 dwt. (53·7 %) respectively; total, 74·52 dwt. (54·8 %); loss, 61·49 dwt. (45·2 %). Recovery of a proportion of the gold by amalgamation effects a saving also in royalty fees which is found to be worth consideration. Combined milling

(dry) and cyaniding is reported to cost 11*s.* 9½*d.* per ton. As the result of long experiment, dry milling is to be replaced by wet, thus obviating expenditure on fuel for drying the ore and doubling (at least) the output of the battery.* The introduction of Frue vanners is also probable, though the amount of concentrates would be only about 1 %, and they require roasting and treatment with 5 % KCy solution as against 1·4 % for the bulk.

**Cyanidation only.**—Descriptions have already been given (pp. 658–84) of plants for cyanidation, and it remains only to add a few examples of complete mills and their working results.

Among New Zealand mills the Waihi is most prominent. In 1898, its costs on 86,696 t. were:—Drying, 1*s.* 2·13*d.*; breaking, 10·86*d.*; milling, 3*s.* 0·65*d.*; cyanidation, 3*s.* 0·9*d.*; melting bullion, 6·96*d.*; total, 8*s.* 9·5*d.*

At the Waitekauri, in 1899, the cost of dry milling and cyaniding 26,407 t. was 14*s.* 10½*d.* per ton. The plant is now being converted from dry to wet stamping.

The new mill of the Moanataiari Co. has two 16 × 10-in. swing-jaw breakers delivering into a bin which fills trucks. These are raised by a hydraulic lift, and the contents are dumped into the battery hoppers. The stamps are 1060 lb., dropping 95 per minute, with an independent cam-shaft to each 5 heads. Automatic feeders are supplied, and the duty is expected to be 4 t. per stamp per 24 hours. The tables are fitted with Muntz-metal plates 10 in. long and each 5 stamps have 2 vanners; 12 being Frues and 12 Unions. The concentrates are trucked to the vat house, hoisted in a hydraulic lift, and dumped into 3 steel vats 20 ft. diam. by 7 ft. deep, for cyaniding. These vats are carried upon trestle-work, and the solutions gravitate to two extraction-boxes for zinc precipitation. The mill is also provided with 21 Berdan pans, which formed part of a previous plant. If found advisable, it is intended to buddle the vanner tailings, and treat the heads in the Berdans. Both water and steam power are

* Official Report for 1899.

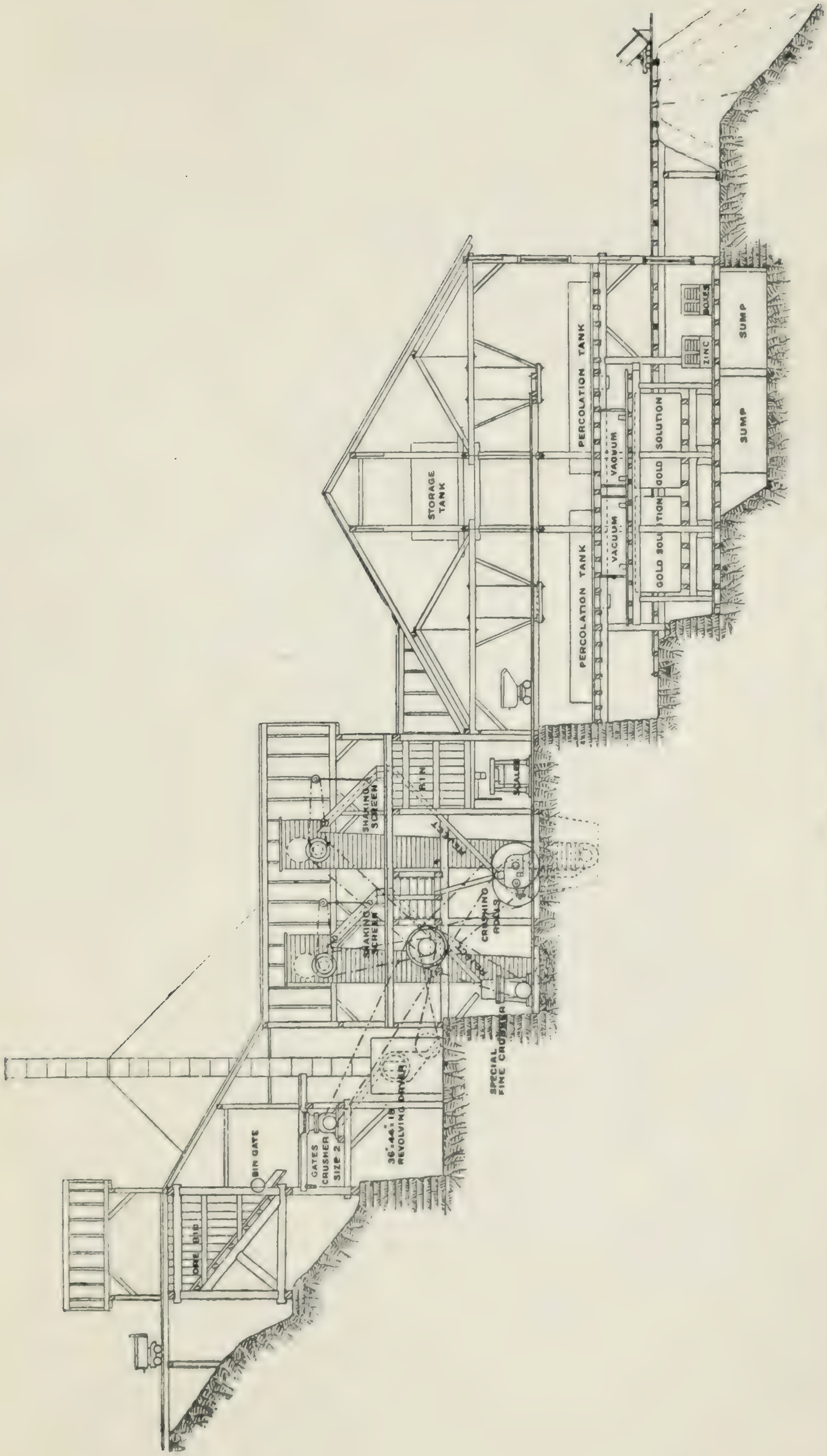



FIG. 205.—CYANIDATION MILL.

provided for use, according to circumstances, the engine being 250 h.p. The hydraulic power plant consists of 9 Pelton wheels, each section having its own motor. The mill is lighted by electricity, and contains a single stamp for crushing specimen stone.*

The costs of dry milling and cyaniding at the Lisbon-Berlyn, Transvaal, in 1899, on 28,555 t. milled (27,457 t.

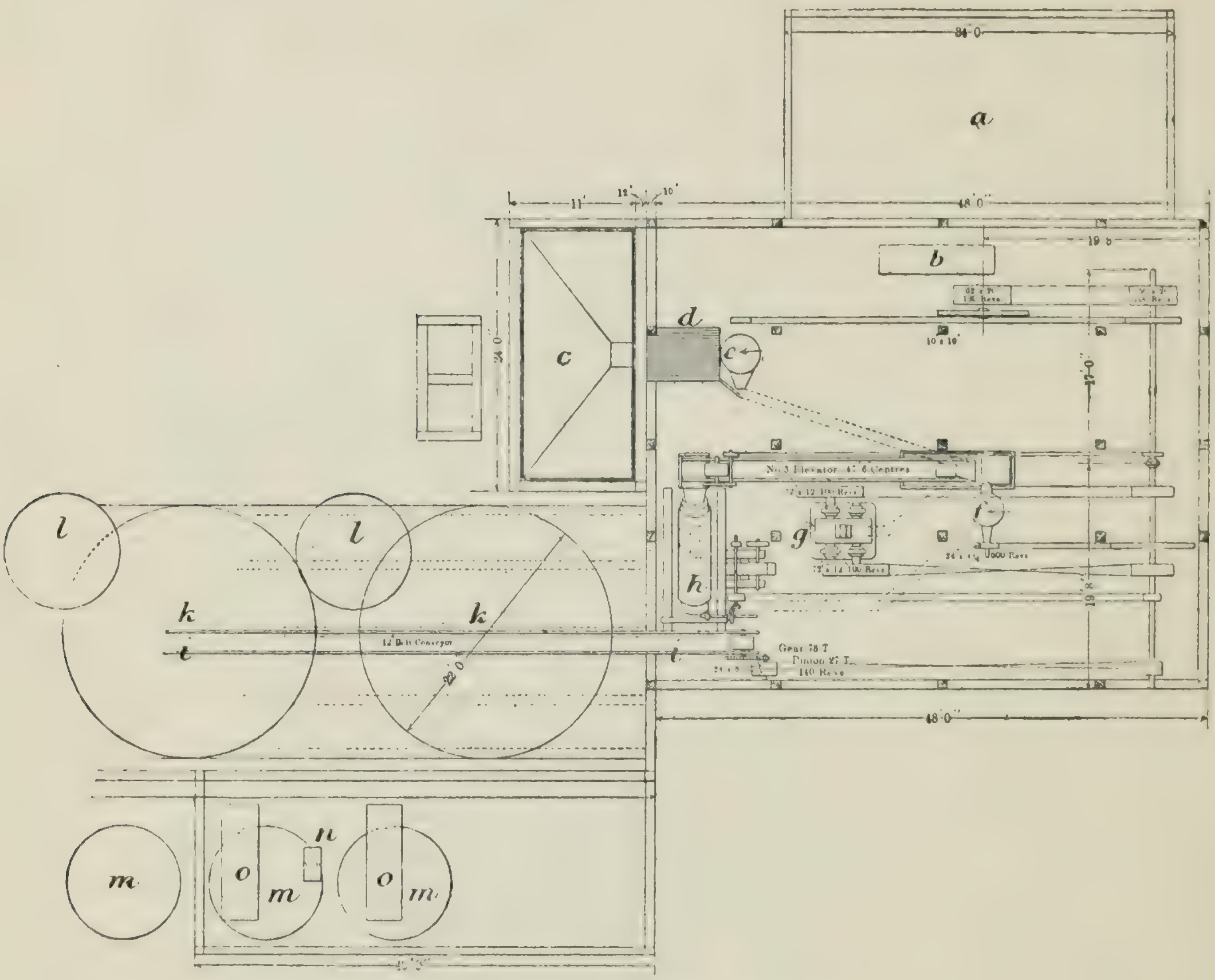



FIG. 207.—CYANIDATION MILL.

cyanided) were—milling, 1*s.* 3·54*d.*; cyaniding, 6*s.* 1·76*d.*; total, 7*s.* 5·3*d.* per ton.

In Fig. 206 is shown a 50-t. cyanide mill, by the Gates Iron Works, receiving dry-crushed ore by truck from a bin and delivering its residues also by truck to the dump.

The plant of the Rose G. M. Co., at Victor, California, illustrated in Figs. 207, 208 is one of the newest and most advanced in the United States. The mill is located over the

* A. H. Bromly, En. & Min. Jl., Nov. 12, 1898.

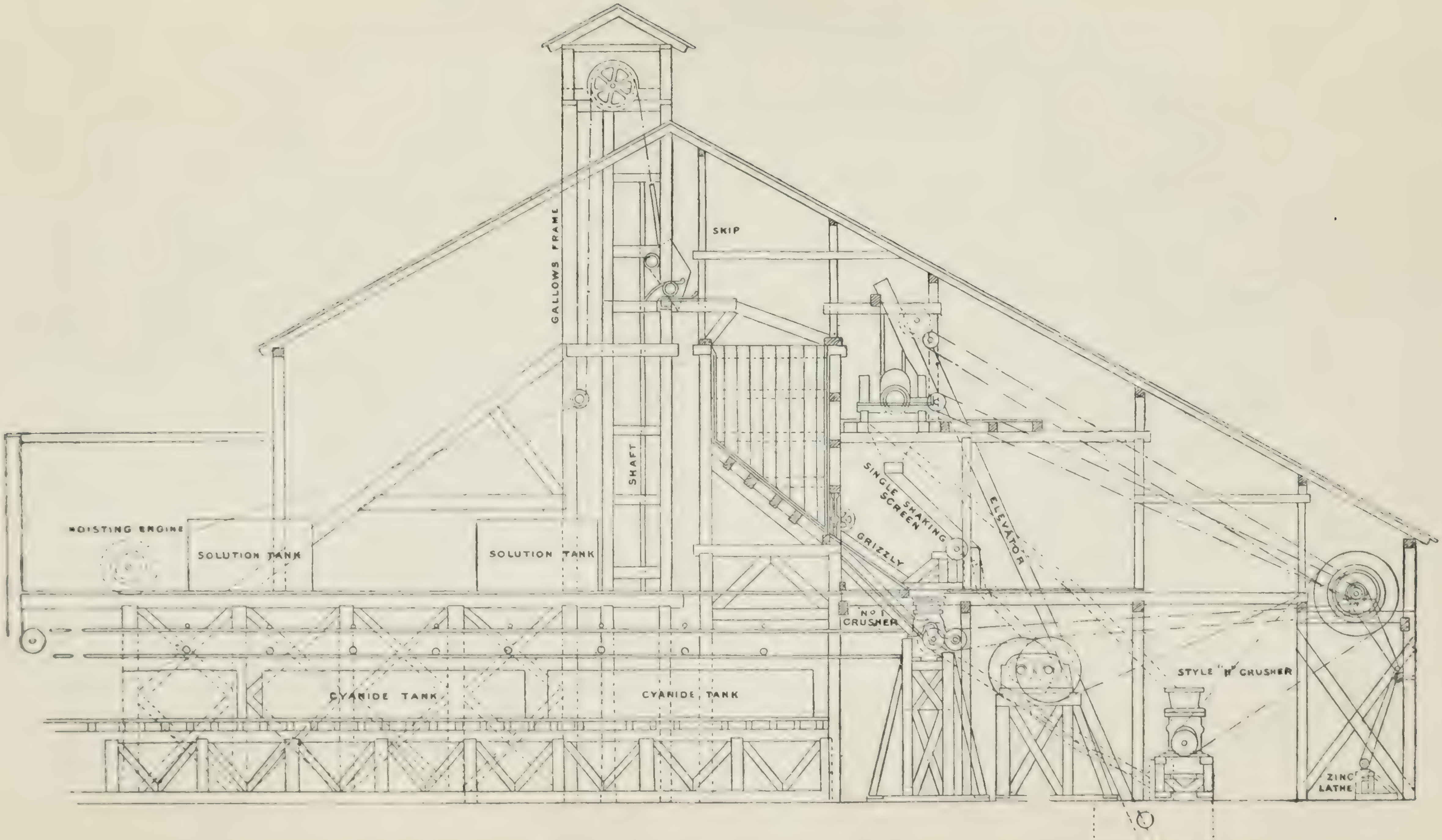



FIG. 208.—CYANIDATION MILL

mine shaft from which the ore is hoisted. The skip is automatically tipped, allowing the ore to fall into a storage bin *c* having a capacity of 75 t. From this it is fed over grizzly bars *d*, 1½ in. apart, to a Gates No. 1 crusher *e*, where it is reduced to 1½ in. and thence conducted to a hopper common to the rolls, the No. 1 crusher and "H" crusher whence it drops into a Gates No. 3 elevator, and is elevated to a 32-in. by 8-ft. revolving screen, having apertures ¾ in. diam.; the rejections from the screen are conducted to a Gates H crusher *f*, which reduces the material to ¾ in. or less. It passes thence to the elevator, which returns it to the revolving screen *h*, the undersize of which passes over a single shaking screen of 3 holes to the inch. The undersize of the shaking screen is finished product, and is conducted to a small storage bin, from which it is automatically fed on a 12-in. belt conveyor *i*, and carried to the leaching vat *k* receiving the charge. The oversize of the shaking screen passes to a set of Gates 26 by 15-in. rolls *g*, and leaving the rolls it is conducted to the elevator, which returns it to the revolving screen. The dumping of the finished product from the conveyor into the leaching vat is effected by means of two false rollers, one set at an angle of 60°, over which the belt passes, thereby raising the belt from a horizontal to an inclined position, and the other roller set at an angle of 50°, under which the belt passes after having discharged its burden. The belt may be set so as to discharge at any one of three points in a vat. When a tank is charged, horizontal rollers are substituted for the false rollers, which are then removed and put in position over the vat next to receive a charge. With wages ruling at 10*s.* 6*d.* to 14*s.* a day, and on an output of 33 t. per 11-hour shift, the working costs are stated by the managing chemist, C. T. Arkins,* as follows:—Labour, 1*s.* 5*d.*; fuel, 4½*d.*; oil, 1*d.*; lime, cyanide, and zinc, 2*s.* 8*d.*; discharging vats, 5*d.*; total, 4*s.* 11½*d.* per ton. The ore is difficult to cyanide, as it contains much clay and talc, and quite a quantity of cupriferous mineral, mostly malachite and azurite. Much trouble is encountered with precipitation, but by the continual use of

* En. & Min. Jl., Jan. 13, 1900.

lead acetate on the zinc turnings, and strengthening the cyanide solution before it enters the boxes, satisfactory precipitation is generally maintained. The gold recovery ranges between 90 and 95 %. Loss in solid residues ranges between ·3 and ·4 dwt. Ore assaying less than 5 dwt. is profitably treated.

At the custom mill of the Brodie Reduction Co., Cripple Creek, the ore as received is unloaded into bins, each lot being kept separate until sampled and paid for. From the bins it is delivered by hand to a Gates breaker, which reduces it to pieces not exceeding 1 in. diam. Thence it is raised by a belt elevator, and passes through a Vezin sampler. The crushed ore passes through a 4-tube Argall drier to a Dodge crusher, and is again lifted in a belt elevator to a revolving screen (iron wire 2-mesh), the oversize passing to Davis rolls, whence it is returned to the screen, and the undersize to Krom rolls (14 × 20 in.). The product of the Krom rolls is elevated, divided, and delivered to 4 revolving screens (40-mesh brass wire cloth). The undersize is carried by a screw conveyor to the storage bins or roasting furnace, as is desired. The oversize passes to another set of Krom rolls whence it is elevated to the 40-mesh screens. The oxidised surface ores pass directly to the storage bins. At one time the unoxidised telluride ores went to the roasting furnace, but in later practice they are first leached raw and the residues are run over Wilfley tables, the concentrates thus collected alone being roasted before second cyanidation. From the bins the ore is drawn into tram-cars, each carload being weighed and dumped into the leaching vats. The vats are circular, and are constructed of No. 8 iron ; each is provided with a false bottom constructed of two layers of wooden slats covered with coco-nut matting and No. 8 duck, and is provided with man-holes for sluicing off the tailings after the charge is leached. The stock solution is stored in paraffin-coated steel tanks, and is kept at proper strength by addition of KCy. The liquor is run on top of the ore and allowed to percolate. Two solutions are used, the strong containing ·5 to ·75 %. Time of treatment varies from 70 to 100 hours, or more,

according to the ore. Strong solution is allowed to percolate for about 50 hours, when the weak solution is added, and afterward the wash-water.

At the Geyser mill, Marion, Utah, according to W. Magenau,* the ore, being essentially shale and clay with occasional bunches of harder limestone, can be efficiently crushed in gyratory breakers—a rougher discharging into a finisher, and a single rotary screen of 1-in. mesh delivers it into a bin, from which it is hand-trucked on a trestle-tram to the leaching vats. A single treatment lasting 60 to 120 hours is all that is given.

The various steps in the operations carried on at the Bi-metallic Extraction Co.'s works are as follows: The ore is delivered at the sampling works from all parts of Cripple Creek, being purchased on the basis of its assay value, less treatment charges (29 to 46*s.* per ton), including freight from mine to works. The ore is crushed to about 1 in. cube, then cut down by means of a series of Vezin samplers and crushing rolls to a sample passing $\frac{1}{4}$-in. mesh, and containing only $2\frac{1}{2}$ % of the original ore. This is further crushed and cut down until the final sample passes 120-mesh. The main part of the ore is taken by conveyors to storage bins having a capacity of 1500 t. The next step is drying, as a preliminary to screening. The two Argall driers have an aggregate capacity of 260 to 300 t. a day. The ore, heated to about 300° F., is delivered to a series of screens and crushers, the system being as follows: the oversize from the coarse screen passes to crushing rolls, while the material passing the screen advances to the next finer screen. This is carried on until the desired fineness is obtained; the oversize passing each time to finer crushing rolls. The peripheral speed of the rolls is governed by the size of material; rolls receiving up to $1\frac{1}{4}$ in. cube are run at 350 to 400 ft. per minute; rolls receiving 15 to 20-mesh for reduction to 40-mesh, 1000 ft. per minute. There are 3 fine-crushing mills. From No. 1 mill, which reduces 100 t. a day of oxidised ore to 40-mesh, the material passes directly to the leaching vats; from Nos. 2

* En. & Min. Jl., July 21, 1900.

and 3, which crush in the aggregate 370 t. a day of pyritic ore to 30-mesh, it goes to the roasters. The crushing mills are situated close to the roasters, and between them are storage bins providing for 800 t. of ore. The Argall tubular roaster is used. The tubes make a rev. in 4·8 minutes. They terminate in a hood, which is furnished along its periphery with a series of holes, through which the ore escapes into a common hopper. These holes are covered with an iron band, except immediately over the ore hopper, to prevent the escape of flames from the furnace. The slow, steady advance of the streams of ore from the delivery end toward the furnace ensures a gradual heat almost to the point of sintering. If the sulphur is reduced (from 2 %) to ·1 %, satisfactory extraction is obtained. The roasted ore is carried by conveyors travelling over water-cooled surfaces to the leaching tanks, which are built of steel, the largest being 50 ft. diam. The filters consist of cane matting supported by a wooden framework, and covered by 10-oz. duck. The residues are sluiced out by bottom discharge doors. Zinc shavings and zinc dust are used for precipitation.

The Golden Gate mill at Mercur, with a capacity of 500 t. a day, is operated throughout by electric power, transmitted from a distance of 35 miles at a potential of 40,000 volts. The mill is built of steel, on a steep hill-side, and is 294 ft. wide and 420 ft. long, with a vertical height of over 400 ft. The loss of power in transmission is barely 5 %. Current is delivered at 12*l.* 10*s.* per h.p. per annum, with a guaranteed minimum of 300 h.p. The transformer house, which contains the apparatus for converting the current from 40,000 volts three-phase to 220 volts two-phase, is located between the shaft-house and the mill. The ore is hoisted from the shaft on an incline to the crusher situated at the top of the hill, a distance of 800 ft. This first, or upper story of the mill, is devoted to the coarse crushing of the ore, and is 25 ft. wide by 75 ft. long, the height being 100 ft. It is divided into 3 floors. The upper floor receives the ore in bins, just as it comes from the shaft. The middle floor contains the crushers, weighing 20 t. each, and having a combined capacity of 1500 t.

daily. They are operated by two 50-h.p. Westinghouse type C motors. From the crusher floor the ore is lowered to the third level of the upper story, where it is stored in bins of 2000 t. capacity. This entire upper story is divided vertically into 3 compartments, for the different kinds of ore—talcy, silicious and arsenical—which are kept separate through the mill until they are finally mixed for the leaching tanks. The second section contains 2 driers having a capacity of 500 t., room being left for another. The ore is worked over the floors of the driers by rabbling devices operated by endless chains, and is there heated for about 2½ hours. In the third section the fine crushing and sizing are carried on. The machinery consists of 4 sets of 26-in. and 3 of 36-in. rolls, with a maximum crushing capacity of 1000 t. per day; Berthelet apparatus for sizing; and a system of elevators for raising the crushed product to the separators,—2 elevators to each of the three different classes of ore, their lift being 60 ft. There are 3 separators for each class, that which is sufficiently fine passing through to the next section, while that which is still coarse is passed through the crushers until fine enough. The pulverised ore is finally collected in a bin holding 2000 t. Under this is a dust-chamber, which is connected directly with the main flue, and is provided with hoppers spaced 12 ft. apart. The floor is so arranged that the dust collects at the mouths of these hoppers to be drawn off into the cars below. The roasting takes place in 4 Browne straight-line furnaces. The arsenical ores are crushed to about 10 mesh and discharged into two of the furnaces. The talcy ores are crushed to 4 mesh, and roasted in the remaining furnaces. The silicious ores do not require roasting, but are emptied directly into the receiving hoppers below the roasters and mixed with the other ores while there, in proper proportions for the leaching tanks. The furnaces used for the arsenical ores have a daily capacity of 75 t. each; those for the talcy ores of about 150 t. each. The furnaces have their roasting hearths directly on the foundations, the calcined ore being elevated to the cooling floors above. The ore is stirred during roasting and advanced by motor-operated cables once each minute. One

man attends to two furnaces. The gases are carried from the furnaces through 6 × 8-ft. flues into the main dust-chamber, which runs from this section to a steel stack 8 ft. diam. and 85 ft. high, located on the hill above the buildings. From the furnace hoppers the ore passes through a tunnel to a point just above the tanks on the seventh section of the incline. This has two floors, the main floor containing 10 tanks each 25 by 50 ft. and 5 ft. deep, and 3 solution tanks 20 ft. diam. and 12 ft. high. The tanks are supported by masonry piers and a retaining wall. The ore from the tunnels is received in cars and run out by hand over the tanks, where it is dumped. The solutions filter through the bottom of the tanks, passing out to the precipitators on the eighth level. When the liquor has been drawn from the tanks, the tailings are carried out on cars which run underneath the tanks to the waste-dumps. In the eighth or lowest section the gold is precipitated in 3 vats 14 ft. diam. by 8 ft. high, containing zinc turnings. The works contain a gas plant (in which water gas is produced for roasting and drying the ore), assay office, refinery and chemical laboratory.

**Recovering Associated Metals.**—Some remarks have already been made (pp. 470–2) on this subject, where the operations are designed to follow chlorination. Other methods deserving of mention are described below.

In the Park-Whitaker process, for treating cupriferous material unsuited for ordinary cyanidation, the ores or concentrates are subjected to a chloridising roast (pages 366–7), after which the soluble copper chlorides remaining are leached out with water, and the copper is recovered by precipitation with iron. After copper salts have been washed out, the solid residues are treated with an alkaline liquor, and then subjected to weak cyanide solution for extraction of the gold and silver.

The isolation and recovery of both metals when gold is associated with antimony still constitute a problem in practical metallurgy. Among recent proposals the two following are worthy of mention, though it is not known to the author whether either has been applied industrially.

When lead is present in auriferous antimony, Haremann proposes to roast the crushed ore with carbon and an alkaline carbonate, thus obtaining a double sulphide of the alkali and antimony which will leach out and leave the gold behind with the lead, so that it can be recovered by simple smelting or cupellation. The antimony is precipitated as sulphide by addition of an acid (preferably sulphuric), which affords an alkaline sulphate, recoverable for future use.

N. W. Edwards has patented (English, No. 15,791 of 1897) the treatment of auriferous and argentiferous antimony ores by lixiviation with a 7 % solution of calcium sulphide (cold) in large vats provided with stirring apparatus, whereby the antimony is dissolved as sulphide and the gold is left in the residue, whence it may be recovered by ordinary methods. The solution is drawn off and treated in closed vessels with carbonic acid obtained by burning lime, whereby the calcium is precipitated as carbonate, which is filtered off. The solution is then treated again with carbonic acid, which precipitates antimony sulphide. The sulphuretted hydrogen which is given off by the decomposition of the double sulphide solution is conducted through milk of lime to regenerate calcium sulphide.

## TABLE A.—DATA FOR CALCULATIONS.

*Weights.*

1 grain (gr.) = ·0648 *gramme* (*grm.*).
1 pennyweight (dwt.) = 24 gr. = 1·5551 *grm.*
1 *grm.* = 15·4323 gr.
1 dram (dr.) = 1·7718 *grm.*
1 oz. (avoir.) = 437·5 gr. = 28·3495 *grm.*
1 oz. (troy) = 480 gr. = 31·1035 *grm.*
1 lb. (avoir.) = 453·6 *grm.* = ·4536 *kilogramme* (*kilo.*) = 7000 gr. = 16 oz. avoir. = 14·58 oz. troy = 1·2153 lb. troy.
1 lb. (troy) = 373·2419 *grm.* = ·8228 lb. avoir.
1 lb. water = ·1 gal. = ·12 U.S. gal. = 27·72 cub. in.
1 *kilo.* = 1000 *grm.* = 2·2046 lb. avoir. = 35·3 oz. avoir.
1 *kilo.* water = 1 *litre* = 61·02 cub. in. = ·22 gal. = 1·7607 pint.
1 oz. per long ton (2240 lb.) = 1 *milligramme* per 32·67 *grm.*
1 oz. per short ton (2000 lb.) = 1 ,, ,, 29·17 *grm.*
1 cwt. = 112 lb. = 50·824 *kilo.*
1 *tonne* = 1000 *kilo.* = 2204·6 lb. = ·9842 long ton = 1·1023 short ton.
1 short ton = 2000 lb. = 14,000,000 gr. = ·9072 *tonne.*
1 short ton water = 200 gal. = 32·07 cub. ft.
1 short ton battery pulp = 23 cub. ft.
1 short ton battery sands (slimes removed) = 26 cub. ft.
1 long ton = 2240 lb. = 32,666 oz. troy = 15,680,000 gr. = 1016·48 *kilo.* = 1·0164 *tonne.*
1 long ton water = 224 gal. = 35·92 cub. ft.
1 long ton battery pulp = 26 cub. ft.
1 long ton battery sands (slimes removed) = 29 cub. ft.

*Measures.*

1 gal. = 10 lb. = ·16 cub. ft. = 277·3 cub. in. = 4·537 *litres* = 1·2 U.S. gal.
1 U.S. gal. = 8·33 lb. = ·133 cub. ft. = 231 cub. in. = 3·8 *litres* = ·83 gal.
1 cub. ft. = 6·23 gal. = 28·375 *litres.*
1 cub. ft. water = 62·35 lb.
1 yd. = ·9143 *metre.*
1 *metre* = 1·0936 yd. = 3·2809 ft. = 39·3708 in.

*Circles and Ovals.*

Circumference = diam. × 3·1416 = radius × 6·2832.
Diameter = circumf. × ·3183.
Radius = circumf. × ·1591.
Area = diam.$^2$ × ·7854 = radius$^2$ × 3·1416 = circumf. × diam. ÷ 4.
Diam. of circle producing area equal to that of given square = side of square × 1·1283.
Contents of vat = length × width (or diam.$^2$ × ·7854) × depth = result in cub. ft. × 6·24 = result in gal.; or, result in gal. = diam.$^2$ × 5 times the depth in ft. − 2%.

*Temperatures.*

° Fahr. = ° Cent. × 9 ÷ 5 + 32; = ° R. × 9 ÷ 4 + 32.
° Cent. = ° Fahr. − 32 × 5 ÷ 9.
° R. = ° Fahr. − 32 × 4 ÷ 9.

These rules apply only to temperatures above zero.

## Table B.—AVOIRDUPOIS AND TROY EQUIVALENTS.

| Avoir. | Troy. | Avoir. | Troy. | Avoir. | Troy. | Avoir. | Troy. |
|---|---|---|---|---|---|---|---|
| oz. | dwt. | oz. | oz. | lb. | oz. | lb. | oz. |
| ¼ = | 4·55 | 9 = | 8·20 | 5 = | 72·91 | 17 = | 247·92 |
| ½ = | 9·11 | 10 = | 9·11 | 6 = | 87·50 | 18 = | 262·50 |
| ¾ = | 13·67 | 11 = | 10·02 | 7 = | 102·08 | 19 = | 277·08 |
| 1 = | 18·23 | 12 = | 10·92 | 8 = | 116·66 | 20 = | 291·64 |
| | oz. | 13 = | 11·84 | 9 = | 131·25 | 21 = | 306·25 |
| 2 = | 1·82 | 14 = | 12·76 | 10 = | 145·82 | 22 = | 320·82 |
| 3 = | 2·73 | 15 = | 13·67 | 11 = | 160·41 | 23 = | 335·41 |
| 4 = | 3·64 | lb. | | 12 = | 175·00 | 24 = | 350·00 |
| 5 = | 4·56 | 1 = | 14·58 | 13 = | 189·58 | 25 = | 364·58 |
| 6 = | 5·46 | 2 = | 29·16 | 14 = | 204·16 | 26 = | 379·16 |
| 7 = | 6·38 | 3 = | 43·75 | 15 = | 218·75 | 27 = | 393·75 |
| 8 = | 7·29 | 4 = | 58·33 | 16 = | 233·33 | 28 = | 408·33 |

## Table C.—GRAINS AND DWT. IN DECIMALS OF 1 OZ.

| gr. | oz. | gr. | oz. | gr. | oz. | dwt. | oz. |
|---|---|---|---|---|---|---|---|
| 1 = | ·0021 | 12 = | ·0250 | 23 = | ·0479 | 10 = | ·50 |
| 2 = | ·0042 | 13 = | ·0271 | 24 = | ·05 | 11 = | ·55 |
| 3 = | ·0062 | 14 = | ·0292 | dwt. | | 12 = | ·60 |
| 4 = | ·0083 | 15 = | ·0312 | 2 = | ·10 | 13 = | ·65 |
| 5 = | ·0104 | 16 = | ·0333 | 3 = | ·15 | 14 = | ·70 |
| 6 = | ·0125 | 17 = | ·0354 | 4 = | ·20 | 15 = | ·75 |
| 7 = | ·0146 | 18 = | ·0375 | 5 = | ·25 | 16 = | ·80 |
| 8 = | ·0167 | 19 = | ·0396 | 6 = | ·30 | 17 = | ·85 |
| 9 = | ·0188 | 20 = | ·0417 | 7 = | ·35 | 18 = | ·90 |
| 10 = | ·0208 | 21 = | ·0438 | 8 = | ·40 | 19 = | ·95 |
| 11 = | ·0229 | 22 = | ·0458 | 9 = | ·45 | 20 = | 1·0 |

## Table D.—LB., QR., AND CWT. IN DECIMALS OF A TON.

| | Short Ton. | Long Ton. | | Short Ton. | Long Ton. | | Short Ton. | Long Ton. |
|---|---|---|---|---|---|---|---|---|
| lb. | | | lb. | | | lb. | | |
| 1 = | ·0004 | ·0004 | 12 = | ·0060 | ·0054 | 23 = | ·0115 | ·0103 |
| 2 = | ·0010 | ·0009 | 13 = | ·0065 | ·0058 | 24 = | ·0120 | ·0107 |
| 3 = | ·0014 | ·0013 | 14 = | ·0069 | ·0062 | 25 = | ·0126 | ·0111 |
| 4 = | ·0020 | ·0018 | 15 = | ·0075 | ·0067 | 26 = | ·0130 | ·0116 |
| 5 = | ·0024 | ·0022 | 16 = | ·0079 | ·0071 | 27 = | ·0135 | ·0121 |
| 6 = | ·0030 | ·0027 | 17 = | ·0085 | ·0076 | qr. | | |
| 7 = | ·0035 | ·0031 | 18 = | ·0089 | ·0080 | 1 = | ·0140 | ·0125 |
| 8 = | ·0039 | ·0035 | 19 = | ·0095 | ·0085 | 2 = | ·0280 | ·025 |
| 9 = | ·0044 | ·0040 | 20 = | ·0099 | ·0089 | 3 = | ·0420 | ·0375 |
| 10 = | ·0050 | ·0045 | 21 = | ·0105 | ·0094 | cwt. | | |
| 11 = | ·0055 | ·0049 | 22 = | ·0109 | ·0098 | 1 = | ·0560 | ·05 |

TABLE E.—GRAINS, DWT. AND OZ. PER SHORT AND LONG TON IN *GRAMMES* PER *TONNE*.

| Per Ton. | *Grm.* per *Tonne.* Short. | *Grm.* per *Tonne.* Long. | Per Ton. | *Grm.* per *Tonne.* Short. | *Grm.* per *Tonne.* Long. |
|---|---|---|---|---|---|
| gr. | | | dwt. | | |
| 1 | ·07 | ·065 | 9 | 15·43 | 13·78 |
| 2 | ·14 | ·13 | 10 | 17·14 | 15·31 |
| 3 | ·21 | ·19 | 11 | 18·86 | 16·84 |
| 4 | ·28 | ·25 | 12 | 20·57 | 18·37 |
| 5 | ·36 | ·32 | 13 | 22·29 | 19·90 |
| 6 | ·43 | ·38 | 14 | 24·00 | 21·43 |
| 7 | ·50 | ·44 | 15 | 25·71 | 22·96 |
| 8 | ·57 | ·51 | 16 | 27·43 | 24·49 |
| 9 | ·64 | ·58 | 17 | 29·14 | 26·02 |
| 10 | ·71 | ·64 | 18 | 30·86 | 27·55 |
| 11 | ·79 | ·70 | 19 | 32·57 | 29·08 |
| 12 | ·86 | ·77 | oz. | | |
| 13 | ·93 | ·83 | 1 | 34·29 | 30·61 |
| 14 | 1·00 | ·89 | 2 | 68·57 | 61·22 |
| 15 | 1·07 | ·96 | 3 | 102·86 | 91·84 |
| 16 | 1·14 | 1·02 | 4 | 137·14 | 122·45 |
| 17 | 1·22 | 1·08 | 5 | 171·43 | 153·06 |
| 18 | 1·29 | 1·16 | 6 | 205·71 | 183·67 |
| 19 | 1·36 | 1·22 | 7 | 240·00 | 214·29 |
| 20 | 1·43 | 1·28 | 8 | 274·29 | 244·90 |
| 21 | 1·50 | 1·34 | 9 | 308·57 | 275·51 |
| 22 | 1·57 | 1·40 | 10 | 342·86 | 306·12 |
| 23 | 1·64 | 1·47 | 20 | 685·71 | 612·24 |
| dwt. | | | 30 | 1028·57 | 918·37 |
| 1 | 1·71 | 1·53 | 40 | 1371·43 | 1224·49 |
| 2 | 3·43 | 3·06 | 50 | 1714·29 | 1530·61 |
| 3 | 5·14 | 4·59 | 60 | 2057·14 | 1836·73 |
| 4 | 6·86 | 6·12 | 70 | 2400·00 | 2142·86 |
| 5 | 8·57 | 7·65 | 80 | 2742·86 | 2448·98 |
| 6 | 10·29 | 9·18 | 90 | 3085·71 | 2755·10 |
| 7 | 12·00 | 10·71 | 100 | 3428·57 | 3061·22 |
| 8 | 13·71 | 12·24 | | | |

TABLE F.—*GRAMMES* PER *TONNE* IN DWT. AND OZ. PER SHORT AND LONG TON.

| *Grm.* per *Tonne.* | Per Short Ton. | Per Long Ton. | *Grm.* per *Tonne.* | Per Short Ton. | Per Long Ton. |
|---|---|---|---|---|---|
| | dwt. | dwt. | | dwt. | dwt. |
| ·1 | ·06 | ·06 | 20 | 11·66 | 13·08 |
| ·2 | ·12 | ·13 | 30 | 17·50 | 19·58 |
| ·3 | ·17 | ·19 | | oz. | oz. |
| ·4 | ·23 | ·26 | 40 | 1·166 | 1·306 |
| ·5 | ·29 | ·32 | 50 | 1·458 | 1·633 |
| ·6 | ·35 | ·39 | 60 | 1·750 | 1·960 |
| ·7 | ·41 | ·45 | 70 | 2·041 | 2·287 |
| ·8 | ·47 | ·52 | 80 | 2·332 | 2·612 |
| ·9 | ·52 | ·59 | 90 | 2·624 | 2·960 |
| 1 | ·58 | ·65 | 100 | 2·916 | 3·266 |
| 2 | 1·16 | 1·30 | 200 | 5·832 | 6·532 |
| 3 | 1·75 | 1·96 | 300 | 8·750 | 9·798 |
| 4 | 2·33 | 2·62 | 400 | 11·664 | 13·064 |
| 5 | 2·91 | 3·25 | 500 | 14·582 | 16·330 |
| 6 | 3·50 | 3·92 | 600 | 17·500 | 19·596 |
| 7 | 4·08 | 4·58 | 700 | 20·418 | 22·862 |
| 8 | 4·66 | 5·24 | 800 | 23·330 | 26·128 |
| 9 | 5·25 | 5·87 | 900 | 26·250 | 29·394 |
| 10 | 5·83 | 6·54 | 1000 | 29·164 | 32·660 |

TABLE G.—CARATAGE AND VALUE PER OZ. OF BULLION ACCORDING TO CONTENTS OF FINE GOLD IN GR. AND DWT.

| Fine Gold. | | | Caratage. | | | Value per oz. | | |
|---|---|---|---|---|---|---|---|---|
| oz. | dwt. | gr. | carats. | gr. | eighths. | £ | s. | d. |
| | | ·625 | | | 1 | | | 1·327 |
| | | 1·250 | | | 2 | | | 2·655 |
| | | 1·875 | | | 3 | | | 3·982 |
| | | 2·500 | | | 4 | | | 5·309 |
| | | 3·125 | | | 5 | | | 6·637 |
| | | 3·750 | | | 6 | | | 7·964 |
| | | 4·375 | | | 7 | | | 9·292 |
| | | 5 | | 1 | | | | 10·619 |
| | | 10 | | 2 | | | 1 | 9·239 |
| | | 15 | | 3 | | | 2 | 7·858 |
| | | 20 | 1 | | | | 3 | 6·477 |
| | 1 | 16 | 2 | | | | 7 | 0·954 |
| | 2 | 12 | 3 | | | | 10 | 7·432 |
| | 3 | 8 | 4 | | | | 14 | 1·909 |
| | 4 | 4 | 5 | | | | 17 | 8·386 |
| | 5 | | 6 | | | 1 | 1 | 2·863 |
| | 5 | 20 | 7 | | | 1 | 4 | 9·341 |
| | 6 | 16 | 8 | | | 1 | 8 | 3·818 |
| | 7 | 12 | 9 | | | 1 | 11 | 10·295 |
| | 8 | 8 | 10 | | | 1 | 15 | 4·773 |
| | 9 | 4 | 11 | | | 1 | 18 | 11·250 |
| | 10 | | 12 | | | 2 | 2 | 5·727 |
| | 10 | 20 | 13 | | | 2 | 6 | 0·204 |
| | 11 | 16 | 14 | | | 2 | 9 | 6·682 |
| | 12 | 12 | 15 | | | 2 | 13 | 1·159 |
| | 13 | 8 | 16 | | | 2 | 16 | 7·636 |
| | 14 | 4 | 17 | | | 3 | 0 | 2·113 |
| | 15 | | 18 | | | 3 | 3 | 8·590 |
| | 15 | 20 | 19 | | | 3 | 7 | 3·068 |
| | 16 | 16 | 20 | | | 3 | 10 | 9·545 |
| | 17 | 12 | 21 | | | 3 | 14 | 4·023 |
| | 18 | 18 | 22 | | | 3 | 17 | 10·500 |
| | 19 | 4 | 23 | | | 4 | 1 | 4·977 |
| 1 | | | 24 | | | 4 | 4 | 11·454 |

TABLE H.—VALUE PER OZ. OF BULLION ACCORDING TO FINENESS (PARTS PER 1000).

| Fineness: parts per 1000. | Value per Oz. | | Fineness: parts per 1000. | Value per Oz. | | | | |
|---|---|---|---|---|---|---|---|---|
| | *d.* | c. | | £ | *s.* | *d.* | $ | c. |
| ·1 | ·10 | ·21 | 20 | | 1 | 8·39 | | 41·34 |
| ·2 | ·20 | ·41 | 30 | | 2 | 6·58 | | 62·02 |
| ·3 | ·31 | ·62 | 40 | | 3 | 4·78 | | 82·69 |
| ·4 | ·41 | ·83 | 50 | | 4 | 2·93 | 1 | 03·36 |
| ·5 | ·51 | 1·03 | 60 | | 5 | 1·17 | 1 | 24·03 |
| ·6 | ·61 | 1·24 | 70 | | 5 | 11·36 | 1 | 44·70 |
| ·7 | ·71 | 1·45 | 80 | | 6 | 9·56 | 1 | 65·38 |
| ·8 | ·82 | 1·65 | 90 | | 7 | 7·75 | 1 | 86·05 |
| ·9 | ·92 | 1·86 | 100 | | 8 | 5·95 | 2 | 06·72 |
| 1 | 1·02 | 2·07 | 200 | | 16 | 11·89 | 4 | 13·44 |
| 2 | 2·04 | 4·13 | 300 | 1 | 5 | 5·84 | 6 | 20·16 |
| 3 | 3·06 | 6·20 | 400 | 1 | 13 | 11·78 | 8 | 26·88 |
| 4 | 4·08 | 8·27 | 500 | 2 | 2 | 5·73 | 10 | 33·61 |
| 5 | 5·10 | 10·34 | 600 | 2 | 10 | 11·67 | 12 | 40·33 |
| 6 | 6·12 | 12·40 | 700 | 2 | 19 | 5·62 | 14 | 47·05 |
| 7 | 7·14 | 14·47 | 800 | 3 | 7 | 11·56 | 16 | 53·77 |
| 8 | 8·16 | 16·54 | 900 | 3 | 16 | 5·50 | 18 | 60·49 |
| 9 | 9·18 | 18·60 | 1000 | 4 | 4 | 11·45 | 20 | 67·21 |
| 10 | 10·19 | 20·67 | | | | | | |

TABLE I.—YIELD PER SHORT TON, LONG TON, AND *TONNE* CALCULATED UPON WEIGHT OF ASSAY PRILL FROM 50-*GRM.* SAMPLE.

| Weight of Prill. | Yield per Short Ton. | | Yield per Long Ton. | | Yield per *Tonne*. |
|---|---|---|---|---|---|
| milligrammes. | oz. | dwt. | oz. | dwt. | grm. |
| ·05 | | ·58 | | ·66 | 1 |
| ·1 | | 1·17 | | 1·31 | 2 |
| ·2 | | 2·33 | | 2·63 | 4 |
| ·3 | | 3·50 | | 3·94 | 6 |
| ·4 | | 4·66 | | 5·26 | 8 |
| ·5 | | 5·83 | | 6·58 | 10 |
| ·6 | | 7·00 | | 7·90 | 12 |
| ·7 | | 8·17 | | 9·14 | 14 |
| ·8 | | 9·33 | | 10·52 | 16 |
| ·9 | | 10·50 | | 11·79 | 18 |
| 1 | | 11·66 | | 13·10 | 20 |
| 2 | 1·166 | | 1·306 | | 40 |
| 3 | 1·749 | | 1·959 | | 60 |
| 4 | 2·332 | | 2·612 | | 80 |
| 5 | 2·915 | | 3·265 | | 100 |
| 6 | 3·500 | | 3·918 | | 120 |
| 7 | 4·081 | | 4·571 | | 140 |
| 8 | 4·664 | | 5·224 | | 160 |
| 9 | 5·250 | | 5·879 | | 180 |
| 10 | 5·830 | | 6·530 | | 200 |

TABLE K.—YIELD PER SHORT AND LONG TON CALCULATED UPON WEIGHT OF ASSAY PRILL FROM 500-GR. SAMPLE.

| Weight of Prill. | Yield per Short Ton. | | Yield per Long Ton. | |
|---|---|---|---|---|
| gr. | oz. | dwt. | oz. | dwt. |
| ·00025 | | ·291 | | ·325 |
| ·0005 | | ·583 | | ·653 |
| ·001 | | 1·166 | | 1·306 |
| ·002 | | 2·333 | | 2·612 |
| ·003 | | 3·500 | | 3·920 |
| ·004 | | 4·666 | | 5·224 |
| ·005 | | 5·833 | | 6·533 |
| ·006 | | 7·000 | | 7·840 |
| ·007 | | 8·166 | | 9·144 |
| ·008 | | 9·333 | | 10·448 |
| ·009 | | 10·499 | | 11·756 |
| ·01 | | 11·666 | | 13·066 |
| ·02 | 1·166 | | 1·306 | |
| ·03 | 1·750 | | 1·960 | |
| ·04 | 2·332 | | 2·616 | |
| ·05 | 2·916 | | 3·260 | |
| ·06 | 3·500 | | 3·920 | |
| ·07 | 4·082 | | 4·574 | |
| ·08 | 4·666 | | 5·232 | |
| ·09 | 5·250 | | 5·888 | |
| ·1 | 5·832 | | 6·520 | |
| ·2 | 11·666 | | 13·066 | |
| ·3 | 17·500 | | 19·585 | |
| ·4 | 23·333 | | 26·133 | |
| ·5 | 29·166 | | 32·666 | |
| ·6 | 35·000 | | 39·200 | |
| ·7 | 40·832 | | 45·733 | |
| ·8 | 46·666 | | 52·266 | |
| ·9 | 52·500 | | 58·800 | |
| 1 | 58·333 | | 65·333 | |

TABLE L.—BULLION CONTENTS PER SHORT AND LONG TON CALCULATED UPON ASSAY PRILL FROM 8 OZ. ($\frac{1}{2}$ LB.) GOLD-CYANIDE SOLUTION.

| Weight of Assay Prill. | Bullion Contents per Short Ton. | | Bullion Contents per Long Ton. | |
|---|---|---|---|---|
| gr. | oz. | dwt. | oz. | dwt. |
| ·001 | | ·16 | | ·19 |
| ·002 | | ·33 | | ·37 |
| ·003 | | ·50 | | ·56 |
| ·004 | | ·66 | | ·75 |
| ·005 | | ·83 | | ·91 |
| ·006 | | 1·00 | | 1·10 |
| ·007 | | 1·16 | | 1·29 |
| ·008 | | 1·33 | | 1·50 |
| ·009 | | 1·50 | | 1·66 |
| ·01 | | 1·66 | | 1·83 |
| ·02 | | 3·33 | | 3·66 |
| ·03 | | 5·00 | | 5·50 |
| ·04 | | 6·66 | | 7·33 |
| ·05 | | 8·33 | | 9·16 |
| ·06 | | 10·00 | | 11·00 |
| ·07 | | 11·66 | | 12·88 |
| ·08 | | 13·33 | | 14·92 |
| 09 | | 15·00 | | 16·65 |
| ·1 | | 16·66 | | 18·33 |
| ·2 | 1·66 | | 1·83 | |
| ·3 | 2·50 | | 2·75 | |
| ·4 | 3·33 | | 3·66 | |
| ·5 | 4·16 | | 4·58 | |
| ·6 | 5·00 | | 5·50 | |
| ·7 | 5·83 | | 6·41 | |
| ·8 | 6·66 | | 7·33 | |
| ·9 | 7·50 | | 8·25 | |
| 1 | 8·33 | | 9·16 | |

TABLE M.—BULLION CONTENTS PER SHORT AND LONG TON CALCULATED UPON ASSAY PRILL FROM 10 OZ. (½ PINT) GOLD-CYANIDE SOLUTION.

| Weight of Assay Prill. | Bullion Contents per Short Ton. | | Bullion Contents per Long Ton. | |
|---|---|---|---|---|
| gr. | oz. | dwt. | oz. | dwt. |
| ·001 | | ·13 | | ·15 |
| ·002 | | ·27 | | ·29 |
| ·003 | | ·40 | | ·44 |
| ·004 | | ·54 | | ·58 |
| ·005 | | ·67 | | ·72 |
| ·006 | | ·80 | | ·89 |
| ·007 | | ·93 | | 1·05 |
| ·008 | | 1·07 | | 1·18 |
| ·009 | | 1·20 | | 1·33 |
| ·01 | | 1·33 | | 1·50 |
| ·02 | | 2·66 | | 3·00 |
| ·03 | | 4·00 | | 4·50 |
| ·04 | | 5·33 | | 6·00 |
| ·05 | | 6·66 | | 7·46 |
| ·06 | | 8·00 | | 9·00 |
| ·07 | | 9·33 | | 10·46 |
| ·08 | | 10·66 | | 11·95 |
| ·09 | | 12·00 | | 13·50 |
| ·1 | | 13·33 | | 14·95 |
| ·2 | 1·33 | | 1·49 | |
| ·3 | 2·00 | | 2·23 | |
| ·4 | 2·66 | | 2·98 | |
| ·5 | 3·33 | | 3·73 | |
| ·6 | 4·00 | | 4·47 | |
| ·7 | 4·66 | | 5·22 | |
| ·8 | 5·33 | | 5·96 | |
| ·9 | 6·00 | | 6·71 | |
| 1 | 6·66 | | 7·45 | |

TABLE N.—BULLION CONTENTS PER 100 GAL. CALCULATED UPON ASSAY PRILL FROM 10 OZ. ($\frac{1}{2}$ PINT) GOLD-CYANIDE SOLUTION.

| Weight of Assay Prill. | Bullion Contents per 100 Gal. | Weight of Assay Prill. | Bullion Contents per 100 Gal. | |
|---|---|---|---|---|
| gr. | dwt. | gr. | oz. | dwt. |
| ·001 | ·06 | ·06 | | 4·00 |
| ·002 | ·13 | ·07 | | 4·66 |
| ·003 | ·20 | ·08 | | 5·33 |
| ·004 | ·26 | ·09 | | 6·00 |
| ·005 | ·33 | ·10 | | 6·66 |
| ·006 | ·40 | ·20 | | 13·33 |
| ·007 | ·42 | ·30 | 1·00 | |
| ·008 | ·53 | ·40 | 1·33 | |
| ·009 | ·60 | ·50 | 1·66 | |
| ·01 | ·66 | ·60 | 2·00 | |
| ·02 | 1·33 | ·70 | 2·33 | |
| ·03 | 2·00 | ·80 | 2·66 | |
| ·04 | 2·66 | ·90 | 3·00 | |
| ·05 | 3·33 | 1 | 3·33 | |

TABLE O.—CAPACITIES OF CIRCULAR VATS IN CUB. FT. ACCORDING TO DIAMETER AND DEPTH, WITHOUT FILTER SPACE OR OTHER DEDUCTION.

| Diam. Ft. | Area Sq. Ft. | Depth. | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 1½ Ft. | 2 Ft. | 3 Ft. | 4 Ft. | 4½ Ft. | 5 Ft. | 5½ Ft. | 6 Ft. | 6½ Ft. | 7 Ft. |
| 7 | 38·48 | 58 | 77 | 115 | 154 | 173 | 192 | 212 | 231 | 250 | 269 |
| 7½ | 44·17 | 66 | 88 | 132 | 177 | 199 | 221 | 243 | 265 | 287 | 309 |
| 8 | 50·26 | 75 | 101 | 151 | 201 | 226 | 251 | 276 | 302 | 327 | 352 |
| 8½ | 56·74 | 85 | 113 | 170 | 227 | 255 | 284 | 312 | 340 | 369 | 397 |
| 9 | 63·61 | 95 | 127 | 191 | 254 | 286 | 318 | 350 | 382 | 413 | 445 |
| 9½ | 70·88 | 106 | 142 | 212 | 284 | 319 | 354 | 390 | 425 | 461 | 496 |
| 10 | 78·53 | 118 | 157 | 236 | 314 | 353 | 393 | 432 | 471 | 510 | 550 |
| 10½ | 86·59 | 130 | 173 | 261 | 346 | 390 | 433 | 476 | 520 | 563 | 606 |
| 11 | 95·03 | 143 | 190 | 285 | 380 | 428 | 475 | 523 | 570 | 618 | 665 |
| 11½ | 103·86 | 156 | 208 | 312 | 415 | 467 | 519 | 571 | 623 | 675 | 727 |
| 12 | 113·09 | 170 | 226 | 339 | 452 | 509 | 565 | 622 | 679 | 735 | 792 |
| 12½ | 122·71 | 184 | 245 | 368 | 491 | 552 | 614 | 675 | 736 | 798 | 859 |
| 13 | 132·73 | 199 | 265 | 398 | 531 | 597 | 664 | 730 | 796 | 863 | 929 |
| 13½ | 143·13 | 215 | 286 | 429 | 573 | 644 | 716 | 787 | 859 | 930 | 1002 |
| 14 | 153·93 | 231 | 308 | 462 | 616 | 693 | 770 | 847 | 924 | 1001 | 1078 |
| 14½ | 165·13 | 248 | 330 | 495 | 661 | 743 | 826 | 908 | 991 | 1073 | 1156 |
| 15 | 176·71 | 265 | 353 | 530 | 707 | 795 | 884 | 972 | 1060 | 1149 | 1237 |
| 15½ | 188·69 | 283 | 377 | 566 | 755 | 849 | 943 | 1038 | 1132 | 1226 | 1321 |
| 16 | 201·06 | 302 | 402 | 603 | 804 | 905 | 1005 | 1106 | 1206 | 1307 | 1407 |
| 16½ | 213·82 | 321 | 427 | 641 | 855 | 962 | 1069 | 1176 | 1283 | 1390 | 1497 |
| 17 | 226·98 | 340 | 454 | 681 | 908 | 1021 | 1135 | 1248 | 1362 | 1475 | 1589 |
| 17½ | 240·52 | 361 | 481 | 722 | 962 | 1082 | 1203 | 1323 | 1443 | 1563 | 1684 |
| 18 | 254·46 | 382 | 509 | 763 | 1018 | 1145 | 1272 | 1478 | 1527 | 1654 | 1781 |
| 18½ | 268·80 | 403 | 538 | 806 | 1075 | 1210 | 1344 | 1400 | 1613 | 1747 | 1882 |
| 19 | 283·52 | 425 | 567 | 851 | 1134 | 1276 | 1418 | 1559 | 1701 | 1843 | 1985 |
| 19½ | 298·64 | 448 | 597 | 896 | 1195 | 1344 | 1493 | 1643 | 1792 | 1941 | 2090 |
| 20 | 314·15 | 471 | 628 | 942 | 1257 | 1414 | 1571 | 1728 | 1885 | 2042 | 2199 |
| 20½ | 330·06 | 495 | 660 | 990 | 1320 | 1485 | 1650 | 1815 | 1980 | 2145 | 2310 |
| 21 | 346·36 | 520 | 693 | 1039 | 1385 | 1559 | 1732 | 1905 | 2078 | 2251 | 2425 |
| 21½ | 363·05 | 545 | 726 | 1089 | 1452 | 1634 | 1815 | 1977 | 2178 | 2360 | 2541 |
| 22 | 380·13 | 570 | 760 | 1140 | 1521 | 1711 | 1901 | 2091 | 2281 | 2471 | 2661 |
| 22½ | 397·60 | 596 | 795 | 1193 | 1590 | 1789 | 1988 | 2187 | 2386 | 2584 | 2783 |
| 23 | 415·47 | 623 | 831 | 1246 | 1662 | 1870 | 2077 | 2286 | 2493 | 2701 | 2908 |
| 23½ | 433·73 | 650 | 867 | 1301 | 1735 | 1952 | 2169 | 2385 | 2602 | 2819 | 3036 |
| 24 | 452·38 | 679 | 905 | 1357 | 1810 | 2036 | 2262 | 2488 | 2714 | 2940 | 3167 |
| 24½ | 471·43 | 707 | 943 | 1414 | 1886 | 2121 | 2357 | 2593 | 2829 | 3064 | 3300 |
| 25 | 490·87 | 736 | 982 | 1472 | 1964 | 2209 | 2454 | 2700 | 2945 | 3191 | 3436 |
| 25½ | 510·70 | 766 | 1021 | 1532 | 2043 | 2298 | 2553 | 2809 | 3064 | 3320 | 3575 |
| 26 | 530·92 | 796 | 1062 | 1593 | 2124 | 2389 | 2655 | 2920 | 3186 | 3451 | 3716 |
| 26½ | 551·54 | 827 | 1103 | 1655 | 2206 | 2482 | 2758 | 3033 | 3309 | 3585 | 3861 |
| 27 | 572·55 | 859 | 1145 | 1717 | 2290 | 2576 | 2863 | 3149 | 3435 | 3722 | 4008 |
| 27½ | 593·95 | 891 | 1188 | 1782 | 2376 | 2673 | 2970 | 3267 | 3564 | 3861 | 4158 |
| 28 | 615·75 | 924 | 1231 | 1848 | 2463 | 2771 | 3079 | 3387 | 3694 | 4002 | 4310 |
| 28½ | 637·93 | 957 | 1276 | 1914 | 2552 | 2871 | 3190 | 3509 | 3828 | 4147 | 4466 |
| 29 | 660·51 | 991 | 1321 | 1982 | 2642 | 2972 | 3303 | 3633 | 3963 | 4293 | 4623 |
| 29½ | 683·49 | 1025 | 1367 | 2050 | 2734 | 3076 | 3417 | 3759 | 4101 | 4443 | 4784 |
| 30 | 706·85 | 1060 | 1414 | 2121 | 2827 | 3181 | 3534 | 3888 | 4241 | 4595 | 4948 |
| 31 | 754·76 | 1132 | 1510 | 2264 | 3019 | 3396 | 3774 | 4151 | 4529 | 4906 | 5283 |
| 32 | 804·24 | 1206 | 1608 | 2413 | 3217 | 3619 | 4021 | 4423 | 4825 | 5228 | 5629 |
| 33 | 855·29 | 1283 | 1711 | 2566 | 3421 | 3849 | 4276 | 4704 | 5132 | 5559 | 5987 |
| 34 | 907·92 | 1362 | 1816 | 2724 | 3632 | 4086 | 4540 | 4994 | 5448 | 5901 | 6355 |
| 35 | 962·11 | 1443 | 1824 | 2886 | 3848 | 4329 | 4811 | 5292 | 5773 | 6254 | 6735 |
| 36 | 1017·87 | 1527 | 2036 | 3054 | 4071 | 4580 | 5089 | 5598 | 6107 | 6616 | 7125 |
| 37 | 1075·21 | 1613 | 2150 | 3225 | 4301 | 4838 | 5376 | 5914 | 6451 | 6989 | 7526 |
| 38 | 1134·11 | 1701 | 2268 | 3402 | 4536 | 5103 | 5671 | 6238 | 6805 | 7372 | 7939 |
| 39 | 1194·59 | 1792 | 2389 | 3583 | 4778 | 5376 | 5973 | 6570 | 7168 | 7765 | 8362 |
| 40 | 1256·63 | 1885 | 2513 | 3770 | 5027 | 5655 | 6283 | 6911 | 7540 | 8168 | 8796 |

TABLE O.—*continued.*

| Diam Ft. | Area Sq. Ft. | Depth. | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 7½ Ft. | 8 Ft. | 8½ Ft. | 9 Ft. | 9½ Ft. | 10 Ft. | 10½ Ft. | 11 Ft. | 11½ Ft. | 12 Ft. |
| 7 | 38·48 | 289 | 308 | 327 | 346 | 365 | 385 | 404 | 423 | 442 | 462 |
| 7½ | 44·17 | 331 | 353 | 375 | 397 | 420 | 442 | 464 | 496 | 518 | 530 |
| 8 | 50·26 | 377 | 402 | 427 | 452 | 477 | 503 | 528 | 553 | 578 | 603 |
| 8½ | 56·74 | 426 | 454 | 482 | 511 | 539 | 567 | 596 | 624 | 652 | 681 |
| 9 | 63·61 | 477 | 509 | 541 | 572 | 604 | 636 | 668 | 700 | 732 | 763 |
| 9½ | 70·88 | 532 | 567 | 602 | 638 | 673 | 709 | 744 | 780 | 815 | 851 |
| 10 | 78·53 | 589 | 628 | 668 | 707 | 746 | 785 | 824 | 864 | 903 | 942 |
| 10½ | 86·59 | 649 | 693 | 736 | 779 | 823 | 866 | 909 | 952 | 995 | 1039 |
| 11 | 95·03 | 713 | 760 | 808 | 855 | 903 | 950 | 997 | 1045 | 1092 | 1140 |
| 11½ | 103·86 | 779 | 831 | 883 | 935 | 987 | 1039 | 1091 | 1142 | 1193 | 1246 |
| 12 | 113·09 | 848 | 905 | 961 | 1018 | 1074 | 1131 | 1187 | 1244 | 1300 | 1357 |
| 12½ | 122·71 | 920 | 982 | 1043 | 1104 | 1166 | 1227 | 1288 | 1350 | 1411 | 1473 |
| 13 | 132·73 | 995 | 1062 | 1128 | 1194 | 1261 | 1327 | 1394 | 1460 | 1526 | 1593 |
| 13½ | 143·13 | 1073 | 1145 | 1217 | 1288 | 1360 | 1431 | 1503 | 1574 | 1645 | 1718 |
| 14 | 153·93 | 1154 | 1231 | 1308 | 1385 | 1462 | 1539 | 1616 | 1693 | 1769 | 1847 |
| 14½ | 165·13 | 1238 | 1321 | 1404 | 1486 | 1569 | 1651 | 1734 | 1816 | 1898 | 1982 |
| 15 | 176·71 | 1325 | 1414 | 1502 | 1590 | 1679 | 1767 | 1855 | 1944 | 2032 | 2121 |
| 15½ | 188·69 | 1415 | 1509 | 1604 | 1698 | 1793 | 1887 | 1981 | 2076 | 2170 | 2264 |
| 16 | 201·06 | 1508 | 1608 | 1709 | 1809 | 1910 | 2011 | 2111 | 2212 | 2312 | 2413 |
| 16½ | 213·82 | 1604 | 1711 | 1817 | 1924 | 2031 | 2138 | 2245 | 2352 | 2459 | 2566 |
| 17 | 226·98 | 1702 | 1816 | 1929 | 2043 | 2156 | 2270 | 2383 | 2497 | 2610 | 2724 |
| 17½ | 240·52 | 1804 | 1924 | 2044 | 2165 | 2285 | 2405 | 2525 | 2646 | 2766 | 2886 |
| 18 | 254·46 | 1908 | 2036 | 2163 | 2290 | 2417 | 2545 | 2672 | 2799 | 2926 | 3054 |
| 18½ | 268·80 | 1016 | 2150 | 2285 | 2419 | 2554 | 2688 | 2822 | 2957 | 3091 | 3226 |
| 19 | 283·52 | 2126 | 2268 | 2410 | 2552 | 2693 | 2835 | 2977 | 3119 | 3260 | 3402 |
| 19½ | 298·64 | 2240 | 2389 | 2538 | 2688 | 2837 | 2986 | 3136 | 3285 | 3434 | 3584 |
| 20 | 314·15 | 2356 | 2513 | 2670 | 2827 | 2984 | 3141 | 3299 | 3456 | 3613 | 3770 |
| 20½ | 330·06 | 2475 | 2640 | 2806 | 2971 | 3136 | 3301 | 3466 | 3631 | 3796 | 3961 |
| 21 | 346·36 | 2598 | 2771 | 2944 | 3117 | 3290 | 3464 | 3637 | 3810 | 3983 | 4156 |
| 21½ | 363·05 | 2723 | 2904 | 3086 | 3267 | 3449 | 3630 | 3812 | 3994 | 4175 | 4357 |
| 22 | 380·13 | 2851 | 3041 | 3231 | 3421 | 3611 | 3801 | 3991 | 4181 | 4371 | 4562 |
| 22½ | 397·60 | 2982 | 3181 | 3380 | 3578 | 3777 | 3976 | 4175 | 4374 | 4572 | 4771 |
| 23 | 415·47 | 3116 | 3324 | 3531 | 3739 | 3947 | 4155 | 4362 | 4570 | 4778 | 4986 |
| 23½ | 433·73 | 3253 | 3470 | 3687 | 3904 | 4120 | 4337 | 4554 | 4771 | 4988 | 5205 |
| 24 | 452·38 | 3393 | 3619 | 3845 | 4071 | 4298 | 4524 | 4750 | 4976 | 5202 | 5429 |
| 24½ | 471·43 | 3536 | 3771 | 4007 | 4243 | 4479 | 4714 | 4950 | 5186 | 5421 | 5657 |
| 25 | 490·87 | 3682 | 3927 | 4172 | 4418 | 4663 | 4909 | 5154 | 5400 | 5645 | 5890 |
| 25½ | 510·70 | 3830 | 4086 | 4341 | 4596 | 4852 | 5107 | 5362 | 5618 | 5873 | 6128 |
| 26 | 530·92 | 3982 | 4247 | 4513 | 4778 | 5044 | 5309 | 5575 | 5840 | 6106 | 6371 |
| 26½ | 551·54 | 4137 | 4412 | 4688 | 4964 | 5239 | 5515 | 5791 | 6067 | 6343 | 6618 |
| 27 | 572·55 | 4294 | 4580 | 4867 | 5153 | 5439 | 5725 | 6012 | 6298 | 6584 | 6871 |
| 27½ | 593·95 | 4455 | 4752 | 5049 | 5346 | 5642 | 5939 | 6236 | 6533 | 6830 | 7127 |
| 28 | 615·75 | 4618 | 4926 | 5234 | 5542 | 5850 | 6157 | 6465 | 6773 | 7081 | 7389 |
| 28½ | 637·93 | 4784 | 5103 | 5422 | 5741 | 6060 | 6379 | 6698 | 7017 | 7336 | 7655 |
| 29 | 660·51 | 4954 | 5284 | 5614 | 5945 | 6275 | 6605 | 6935 | 7266 | 7596 | 7926 |
| 29½ | 683·49 | 5126 | 5468 | 5810 | 6151 | 6493 | 6835 | 7177 | 7518 | 7860 | 8202 |
| 30 | 706·85 | 5301 | 5655 | 6008 | 6362 | 6715 | 7068 | 7422 | 7775 | 8129 | 8482 |
| 31 | 754·76 | 5661 | 6038 | 6415 | 6793 | 7170 | 7548 | 7925 | 8302 | 8680 | 9057 |
| 32 | 804·24 | 6032 | 6434 | 6836 | 7238 | 7640 | 8042 | 8444 | 8847 | 9249 | 9651 |
| 33 | 855·29 | 6415 | 6842 | 7270 | 7698 | 8125 | 8553 | 8981 | 9408 | 9836 | 10263 |
| 34 | 907·92 | 6809 | 7263 | 7717 | 8171 | 8625 | 9079 | 9533 | 9987 | 10441 | 10895 |
| 35 | 962·11 | 7216 | 7697 | 8178 | 8659 | 9140 | 9621 | 10102 | 10583 | 11064 | 11545 |
| 36 | 1017·87 | 7634 | 8143 | 8652 | 9161 | 9670 | 10179 | 10688 | 11197 | 11706 | 12214 |
| 37 | 1075·21 | 8064 | 8602 | 9140 | 9677 | 10214 | 10752 | 11290 | 11827 | 12365 | 12903 |
| 38 | 1134·11 | 8506 | 9073 | 9639 | 10207 | 10774 | 11341 | 11908 | 12475 | 13042 | 13609 |
| 39 | 1194·59 | 8959 | 9557 | 10154 | 10751 | 11349 | 11946 | 12543 | 13140 | 13738 | 14335 |
| 40 | 1256·63 | 9425 | 10053 | 10681 | 11310 | 11938 | 12566 | 13195 | 13823 | 14451 | 15080 |

## Form A.—Return Check of Picked Ore sent up from Mine Lockers.

Co., Ld.

19

*Delivered from Lockers this day Picked Ore as follows:—*

| | | | |
|---|---|---|---|
| Bag No....... | .......lb. | Bag No....... | .......lb. |
| Bag No....... | .......lb. | Bag No....... | .......lb. |
| Bag No....... | .......lb. | Bag No....... | .......lb. |
| Bag No....... | .......lb. | Bag No....... | .......lb. |
| Bag No...... | .......lb. | Bag No....... | .......lb. |
| Bag No....... | .......lb. | Bag No....... | .......lb. |
| Bag No....... | .......lb. | Bag No....... | .......lb. |
| Bag No....... | .......lb. | Bag No....... | .......lb. |
| Total............ | Bags........... | | .......lb. |

Nett Weight, 1 lb. each being allowed for empty bags.............................lb.

Signature...............................Mine Manager.

Checked

.......................................Battery Manager.

## Form B.—Daily Battery Ticket.

| | | | |
|---|---|---|---|
| | | | ............ Co., Ld. |
| | | | ............ 19...... |
| Lost Time . . . . . | | ............ h. | ............ m. |
| Trucks into Bins . . . . | | ............ | |
| Assays : Concentrates ............ oz. | | Tailings ............ oz. | |
| Table Plate Amalgam | ............ balls | | ............ lb. |
| Sluice Plate do. | ............ do. | | ............ lb. |
| Inside Plate do. | ............ do. | | ............ lb. |
| ...... do. | ............ do. | | ............ lb. |
| Total do. | ............ do. | | ............ lb. |
| Crude Amalgam ............ lb. | | | |
| | | | ............ Manager. |

*Note.*—Lost time is calculated to 1 stamp.
Weight of amalgam is returned in lb. avoir.
Crude amalgam is properly allocated when cleaned and re-weighed.

# Form C.—Weekly Battery Ticket.

Co., Ld.

MILL STATEMENT for Week ending 19

Ore Milled ........................... trucks = tons

Time Lost ........................... hours ........... minutes for whole mill

Duty per Stamp per 24 Hours' Actual Work ........................ tons

Amalgam Collected, Crude oz. Clean .................. oz.

Concentrates Collected trucks = ........................ tons

Average Assay of Tailings, in Bullion oz. per ton

Do. Concentrates do. oz. per ton

Concentrates Shipped :—

Nett Weight. Containing oz.

Tons Gold

................................ Silver

Bullion

Picked Ore Received :—

Bags

Containing oz.

Picked Ore Shipped :—

Nett Weight.

Gold

...Silver

Tons .Bullion

Amalgam Retorted oz.

Gold Sponge Produced ....oz.

Melted Gold Produced oz.

Mercury Loss ............ lb.

Manager.

## Form D.—Bullion Deposit Note.

MEMORANDUM OF OUT-TURN OF BULLION LEFT AT..................................................

FOR COINAGE BY.................................................................Co., Ld.

........................................19........, and payable...........................................................................

| Weight before Melting. Oz. | Weight after Melting. Oz. | Assay Report (Decimal). | | Standard Gold. Oz. | Fine Silver Allowed. Oz. | Value. | | |
|---|---|---|---|---|---|---|---|---|
| | | Gold. | Silver. | | | £ | s. | d. |
| | | | | | | | | |

GROSS VALUE { Gold, at 77s. 10½d. per oz. Standard . . .
Silver at......................per oz. Fine . . .

Oz.

Fine Gold

Fine Silver

Fine Bullion

Deduct Charges . . . .

Nett Amount due to Depositor .

# Form E.—Concentrates Shipping Note.

..................................................................Co., Ld.

---

Particulars of Shipment of Lot..................................consisting of .................................. bags ................................ casks Concentrates, despatched from mill.............................................19........, per rail to ................................................Station, S.S. ...............................................

to..........................................being portion of product for............weeks ending..............................................19........

| Gross Weight. | Tare. | Nett Weight. |
|---|---|---|
| ..........................tons. | ..........................lb. | ..........................tons. |

Moisture............................%

Assay per ton: Gold........................oz. Silver........................oz.

Total Contents: Gold........................oz. Silver........................oz.

Valued at £.............................

..........................................................................

Manager.

## Form G.—Dry Milling and Chlorination Return.

..................................................................Co., Ld.........

STATEMENT for Week (Month) ending ............................19........

| | | |
|---|---|---|
| Raw Ore Crushed . . . . . | | ....................tons |
| Concentrates Received . . . | | ....................tons |
| Raw Ore (Concentrates) Roasted. . | | ....................tons |
| Ratio to Ore Milled . . . . | | ....................% |
| Assay Value per ton . . . | Gold | ....................oz. |
| | Silver | ....................oz. |
| | Bullion | ....................oz. |
| Roasted Ore subjected to Chlorination | | ....................tons |
| Ratio to Ore Milled . . . . | | ....................% |
| Assay Value per ton . . . | Gold | ....................oz. |
| | Silver | ....................oz. |
| | Bullion | ....................oz. |
| Gold Recovered per ton . . . | | ....................oz. |
| Silver Lost per ton....................oz. ; | Recovered | ....................oz. |
| Bullion Recovered per ton . . . | | ....................oz. |
| Assay of Residues, per ton . . | Gold | ....................oz. |
| | Silver | ....................oz. |
| | Bullion | ....................oz. |
| Ratio of Extraction . . . | Gold | ....................% |
| | Silver | ....................% |
| | Bullion | ....................% |
| Total Actual Recovery . . | Gold | ....................oz. |
| | Silver | ....................oz. |
| | Bullion | ....................oz. |

..............................................................

Manager.

## Form F.—Monthly Battery Report.

................................................................ Co., Ltd.

TOTALS.

Total yield from ord'y ore, fine gold ............ oz. Total yield per ton ordinary ore, fine gold ............ oz.
do. silver ............ oz. do. silver ............ oz.
do. bullion ............ oz. do. bullion ............ oz.

Loss in tailings per ton ordinary ore, in bullion ........................ oz.
Total contents per ton ordinary ore do. ........................ oz.
Percental recovery per ton ordinary ore do. ........................ %
Picked ore shipped ........................ tons nett.
Picked ore, estimated contents (after treatment in battery), fine gold ........................ oz.
do. silver ........................ oz.
do. bullion ........................ oz.

Estimated grand total product for the month, fine gold ........................ oz.
do. silver ........................ oz.
do. bullion ........................ oz.

Estimated average contents per ton of total product, bullion ........ oz.
Estimated value per ton of total product at £ ..........* per oz. bullion, £ : :
Remarks :

.............................................. Manager.

* According to fineness.

[Face page 786.

# Form F.—Monthly Battery Report.

.............................. Co., Ltd.

MILL STATEMENT for Month ( ........ Weeks) ending .................. 19......

Ore Milled, ordinary .................. trucks, = tons.

Do. picked ........ bags, = ................ tons.

Time lost .................. days .................. hours .......... minutes for .......... stamps.

Duty per stamp for 24 hours' actual work on ordinary ore .................. ton.

Mercury consumed per 100 tons milled, including picked ore ................ lb.

## BATTERY PRODUCTS.

Amalgam retorted .......... oz., being ................ oz. from ordinary and ............ oz. from picked ore.

Crude bullion after retorting .................. oz.

Do. do. do. melting .................. oz.

Do. do. credited by Mint .......... oz.

Fine gold contents : ordinary ore .................. oz. ; special ore .................. oz.

Do. silver do. do. .................. oz. ; do. .......... oz.

Do. bullion do. do. oz. ; do. ... oz.

Total fine bullion contents .................. oz.

Loss of bullion in refining .................. oz.

Standard gold yielded .............. oz. at ........ £ : :

Fine silver allowed ...................... oz. at ............ £ : :

Average yield per ton : ord'y ( ............ tons), fine gold ........ oz. ; picked ( .......... tons), ........ oz.

Do. do. do. do. do. silver ........ oz. , do. ........ oz.

Do. do. do. do. do. bullion ........ oz. ; do. ........ oz.

## VANNER PRODUCTS.

Concentrates, percentage .................. %

Do. shipped, ...................... tons, nett.

Do. average daily assay in bullion per ton ........................ oz.

Do. estimated contents, fine gold ........................ oz.

do. silver ...................... oz.

do. bullion ........ oz.

Do. estimated yield per ton milled (ordinary), fine gold oz.

do. silver .................... oz.

do. bullion .................... oz.

## TOTALS.

Total yield from ord'y ore, fine gold ........ oz. Total yield per ton ordinary ore, fine gold .. oz.

do. silver ........ oz. do. silver .. oz.

do. bullion ................ oz. do. bullion .......... oz.

Loss in tailings per ton ordinary ore, in bullion ...................... oz.

Total contents per ton ordinary ore do. ...................... oz.

Percental recovery per ton ordinary ore do. ...................... %

Picked ore shipped ........................ tons nett.

Picked ore, estimated contents (after treatment in battery), fine gold .............. oz.

do. silver .................. oz.

do. bullion ................ oz.

Estimated grand total product for the month, fine gold oz.

do. silver .............................. oz.

do. bullion .................................. oz.

Estimated average contents per ton of total product, bullion oz.

Estimated value per ton of total product at £ ............* per oz. bullion, £ : :

Remarks :

.............................................. ........ Manager.

* According to fineness.

[Face page 786.

# Form H.—Dry Milling and Cyanidation Return.

.................................................................Co., Ltd.

STATEMENT for Week (Month) ending ........................ 19.......

| | Raw Ore. | Roasted Ore. | Mixed Tailings. | Sands. | Slimes. | Concentrates. | Total. |
|---|---|---|---|---|---|---|---|
| Milled, tons . | | | | | | | |
| Treated, tons . | | | | | | | |
| % of Ore Milled | | | | | | | |
| Assay per ton, oz. | | | | | | | |
| Gold . | | | | | | | |
| Silver . | | | | | | | |
| Bullion . | | | | | | | |
| Recovery per ton treated, oz. | | | | | | | |
| Gold . | | | | | | | |
| Silver . | | | | | | | |
| Bullion . | | | | | | | |
| Assay of Residues per ton, | | | | | | | |
| Gold oz. | | | | | | | |
| Silver . | | | | | | | |
| Bullion . | | | | | | | |
| Ratio of Extraction, . . % | | | | | | | |
| Gold . | | | | | | | |
| Silver . | | | | | | | |
| Bullion . | | | | | | | |
| Product, oz. | | | | | | | |
| Gold . | | | | | | | |
| Silver . | | | | | | | |
| Bullion . | | | | | | | |

.............................

Manager.

## Form I.—Statement of Total Product.

..............................................................Co., Ltd.

SUMMARY OF PRODUCT

For Week (Month) ending ....................................19........

| | Tons. | Ratio to tonnage milled. | Recovery per ton milled. | | | Product. | | | Value. | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Gold. | Silver. | Bullion. | Gold. | Silver. | Bullion. | | | |
| | | % | oz. | oz. | oz. | oz. | oz. | oz. | £ | s. | d. |
| Battery . . | | | | | | | | | | | |
| Concentrates . | | | | | | | | | | | |
| Chlorination . | | | | | | | | | | | |
| Cyanidation . | | | | | | | | | | | |
| ,, Sands | | | | | | | | | | | |
| ,, Slimes | | | | | | | | | | | |
| ,, Concent. | | | | | | | | | | | |
| Other Sources | | | | | | | | | | | |
| Totals . | | | | | | | | | | | |

.............................................................

Manager.

# POSTSCRIPT.

## Amalgamation.

*Page 214.*—The tendency to float exhibited by fine particles of metal, including gold, besides being due in part to their flattened shape, to the attachment of air-bubbles, and to exceptional density of the water caused by impurity, is partially attributed by some investigators to the surface tension of water at rest. It is further remarked that this surface tension is much increased by the addition of oily or greasy matter, and this is offered as one explanation of the results attained by the Elmore oil concentration process (see p. 352). On the other hand, the surface tension is very markedly diminished by matters going into solution, and it is claimed that this fact explains why native gold-washers in many parts of the world make use of wood-lye, the juice of saponaceous plants, urine, etc., when panning down fine gold. It is also in a measure to this fact that Sulman * attributes the efficacy possessed by his soap emulsion in slime settlement.

On the other hand, the detrimental influence of grease in amalgamation, under certain conditions, is most strongly controverted by G. E. Collins,† who maintains that the evil must in all cases be much more fancied than real, inasmuch as an allowance of $\frac{1}{2}$ oz. grease per ton derived from miners' candle-ends would be a conservative estimate for average milling dirt; and he instances some mills at Black Hawk continuing to use filthy water from other mills, even when a pure supply is available (see p. 211). Moreover, he reports a series of

* Trans. Inst. M. & M., viii. 422 (1900).

† Ibid, viii. 423 5 (1900).

drastic experiments made by himself, A. L. Collins, and W. J. Lewis, on the effects produced by deliberate feeding of oil and grease into a battery. The principal trials were conducted in Colorado on ores carrying 10 % of sulphides, and the results of three tests are recorded; these are claimed to indicate that only a very partial injury is caused by grease when the ore is highly mineralised. This is very probably correct, inasmuch as the free acids or soluble sulphates present in such an ore would very greatly counteract the effects of grease. But ores containing 10 % of sulphurets are the exception and not the rule; moreover, the figures relied on for proof of the argument are perhaps not too conclusive. They show that, while the one battery treated with oil afforded 10·4, 12·0, and 12·5 oz. amalgam in the three tests, the average yields from 11 others operating simultaneously on identical ore were 12·6, 12·8, and 13·5 oz. respectively, which proves that the losses amounted to 17·46, 6·25, and 7·4 %—by no means a matter to be ignored.

*Page 259.*—The discussion on Davey's paper * elicited a further statement from him as to the use of sulphuric acid in the amalgamating pan for some ores. Thus he found that the Dark River (Vic.) ore, carrying much mispickel besides 4 % zinc, 3 to 4 % galena, and 2 to 3 % copper, with 1 oz. gold and 10 to 12 oz. silver per ton, would not amalgamate till weak sulphuric acid was added, though lime and caustic soda had been applied. With the help of sulphuric acid he got an extraction of 95 %. The same results were experienced with Bethanga ores, which are of somewhat similar character, the recovery reaching 92 %. The cost is given at 18*s.* per long ton (when operating with 4 Wheeler pans), including roasting; furnacemen receive 1*s.* per hour, and wood is 10*s.* 6*d.* a cord at the furnace. Considerable discretion is necessary in the addition of bichromate and acid, and continually-repeated tests must be made: when the mercury appears globular, there is too much acid; when sluggish, too much bichromate. Without doubting that in Davey's hands good work has been done with the methods he advocates, one cannot help

* Trans. Inst. M. & M., viii. 482 (1900).

wondering how, in the presence of added lime and caustic soda, the acid can have any effect on the constituents of the ore; and why, with an extraction of 92 % at Bethanga, amalgamation should have been abandoned and chlorination (see p. 431) adopted in its stead, for a recovery of 90 %.

### Roasting.

*Page 409.*—The following additional particulars of cost are submitted.

At Cripple Creek, 160 lb. slack coal (worth 8*s.* per ton) are consumed on each ton of ore roasted, and cost 7¾*d.* per ton (8½*d.* per long ton).

At the Great Boulder Main Reef, 1 t. of wood fuel costing 9*s.* per cord suffices for roasting 6 t. (long) of ore, which works out at 9·2*d.* per short ton of ore roasted. Assuming 22*s.* 6*d.* per cord or 13*s.* per ton as the price of future contracts for firewood, the cost of roasting fuel will be about 1*s.* 11·2*d.* per ton of ore furnaced. Present costs of roasting at some West Australian mills are officially said to be entirely covered by less than 4*s.* a ton.

Cost of wear of ironwork in mechanical furnaces is said not to exceed 1*d.* per ton roasted, but presumably this does not include the whole item of maintenance.

### Cyanidation.

*Pyritic Ore, p. 572.*—Sulman * emphasises a fact which is too often overlooked: that many pyritic ores are much more complicated than they seem, and carry minute quantities of antimony, bismuth, molybdenum, vanadium, etc., which are sources of much trouble. In some of the West Australian ores he has found vanadates of lead and mercury. It must be borne in mind, too, that the tellurium in them is much in excess of the gold present, and that about a third of the gold only is in combination, the remainder being free though enveloped perhaps in pyrites. The proportion of tellurium

* Trans. Inst. M. & M., viii. 512 (1900).

accords very closely with that encountered at Cripple Creek, Colorado, where chlorination is very successfully applied; but the calcite in the West Australian ores prohibits that process.

At Hannan's Star it has been found possible to concentrate 40 % of the total value of an ore under 1 oz. into 4 % of the original bulk, and Trewartha-James† attaches much importance to possible developments of concentration methods in the future; but the advantages to be gained from it beyond catching the heavy coarse gold are not very apparent, as the remaining 96 % of the bulk must still be treated for the residual 60 % of value.

The occurrence of very rich tellurides scattered in splashes throughout most of the ore has been observed by A. C. Claudet,‡ some containing 19 % gold and 60 % tellurium. Such an ore would obviously be very bad to roast, and, as with rich concentrates, might be better smelted at once than prepared for any leaching process. Claudet is opposed to any roasting for these ores, if it can be avoided, because of the complications introduced; if carried to the point of decomposition of the basic salts, a simple water-wash may not leach them out.

It is a curious fact that while telluride of gold is quite insoluble in plain potassium cyanide liquor, it is readily attacked by bromo-cyanogen. The chemistry of the reactions involved has so far baffled elucidation; they are probably very complicated, and it would seem that hydrolytic action plays a part, producing tellurous acid (in the form of soluble tellurites) and hydrocyanic acid. It would naturally be inferred that these tellurium salts in solution would give much trouble in precipitation; but, as this has not been found to occur in practice, it is assumed that some compensating influence is at work, by which they are gradually eliminated, in the same way as zinc is (as soluble sulphide) from similar solutions.

The application of bromo-cyanidation to the raw ores at Hannan's Star and the Golden Horseshoe continues to evoke favourable reports. At the former, official statements show

* Trans. Inst. M. & M., viii. 500 (1900).

† Ibid, viii. 506 (1900).

an extraction of 93·75 % on ore carrying 14·09 dwt. per ton, with a residual loss of only ·558 dwt. per ton. At the latter, J K. Wilson uses the bromo-cyanide solution preferably under pressure (in a filter-press) to secure rapidity, because of its tendency to decomposition ; on oxidised ore, the extraction is completed in 20 min. Total cost of filter-pressing (double treatment) slimes is stated at 6*s*. 6*d*. per long ton.

*Precipitation, p. 639.*—Another quite recent variety of the electro-chemical amalgamation idea is the so-called Riecken process, for which the usual extravagant claims of perfection are made,* based on the practically worthless evidence of results obtained in London laboratories. It is not made clear how, in this case more than in the numerous others which have preceded it, the action of cyanide on tellurides is in any way hastened or augmented, or how the drawbacks to cyano-amalgamation already referred to are overcome. The estimates of costs can only be fanciful, and most people will prefer to await the practical results to be achieved by the "apparatus shipped to Western Australia, which will treat 15 tons of ore at a charge."

*Working Results, p. 665. Borneo.*—It will come as a surprise to many that cyanidation plants on a very large scale are being most successfully run in Borneo, and much interest will I am sure be taken in the subjoined notes on the subject, which Mr. A. C. Claudet has most kindly taken the trouble to contribute.

Near Kuching, in the district of Sarawak, a State under the rule of Rajah Brooke and protected by the British Government, the Borneo Co., Ltd., are working auriferous deposits of large extent, for the treatment of which they have erected two mills, about 4 miles apart, at places named Bau and Bidi, the former treating 100,000 t. per annum and the latter 50,000 t. It is reliably estimated that the quantity of ore in the two deposits which can be worked by quarrying alone is in the neighbourhood of 2 million tons, so that there is abundant justification for laying out extensive works. The main characteristics of the ore are alike in the two deposits. The

* H. R. Cassel, En. & Min. Jl., Dec. 8, 1900.

gangue amounts to about 91 %, and contains carbonate of lime and oxide of iron, with more clay at Bau than at Bidi. The ore is essentially oxidised, carrying a little antimony, copper, lead and zinc, together with some arsenic and sulphur, each of these constituents being present in the proportion of less than ·2 %; antimony and arsenic are sensibly more prevalent at Bidi than at Bau. Much the greater part of the gold present is in a remarkably fine state of division, and is not amenable to economic extraction by any process but cyaniding, according to most extensive trials. But its adaptability to cyanide treatment on the most simple lines has been amply proved by Mr. Claudet's investigations. The process adopted at both works, though not absolutely identical in the detail of the milling, otherwise consists uniformly in coarse dry crushing and direct cyanidation with weak solutions. The Bau ore is somewhat argillaceous, and, after passing through a gyratory breaker, is kiln-dried before it goes to the finer crushing in Dodge breakers and rolls, whence it is delivered at a gauge of ½ in. At one time the mesh preferred was 1 in., but latterly the ore has given better results at ½ in. The Bidi ore is crushed no finer than to 2-in. mesh, the only machinery used being Blake-Marsden breakers arranged in two tiers. This is surely the coarsest crushing which has ever been attempted for a lixiviation process, and is in itself a feature which invests the installation with special importance. The adoption of such coarse milling being the outcome of many trials, and not a mere casual circumstance, it possesses much significance, and is distinctly suggestive in relation to similar ores elsewhere. At both works the crushing machinery delivers direct to the leaching tanks, which have 400 t. capacity at Bau and 100 t. at Bidi. In the former case, 2 to 4 lb. lime per long ton of ore are added when charging the vats, being found to accelerate percolation ; at the latter, no lime is needed. The cyanide liquor used in each instance is ·05 to ·075 %, that strength giving maximum extraction ; stronger solutions have been proved to act detrimentally by attacking the base metals in the ore, leading to poorer recovery of gold and greater waste of reagent. The

assay value of the ores ranges between 6 and 8 dwt. per long ton, and the actual extraction is 70 to 80 %, with a cyanide consumption of under 1 lb. per ton. Precipitation is effected in the ordinary zinc-box, and is almost absolute. The total working costs are 1*s.* per long ton for quarrying and delivering to the mills, and 4*s.* for crushing, cyaniding, and all other charges.

**Matte Smelting.**

*Page 703.*—The temperature generated by the smelting of iron protosulphide in Rio Tinto ore was calculated by Hollway at about 4000° F.; Prof. Ackerman reckoned that with an ore carrying 48 % sulphur, 42 % iron and 10% foreign matter, the heat of combustion would be about 3900° F., and that the losses would be 625° in decomposition, 165° in vaporising sulphur, 315° in other gases, and 585° in slag, or a total loss of 1690°, leaving over 2200° available ; Austin computes the maximum temperature attained in pyritic smelting with a blast heated to 750° F. at 4650° F.; and Prof. Howe endorses records of pyrometrical observations showing maximum rises of temperature to points ranging between 2370° and 2530° F.*

*Ores, p. 706.*—At Tilt Cove, Newfoundland, true pyritic smelting, without any extraneous fuel whatever, was for some time practised on lump pyrites carrying 3 to 4 % copper, producing an 8 % matte, with cold blast.

*Furnace, p. 709.*—At Mount Lyell, the blast is heated to about 600° F.

*Results, p. 710.*—The gold losses experienced at Mount Lyell amount in all to about 6½ % on ore assaying 18½ dwt. per long ton. The figures quoted in line 17 on that page refer only to the deficiency recorded in the final treatment of the matte in the converter.

* J. Douglas, Trans. Inst. M. & M., viii. 35 (1900).

# INDEX.

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